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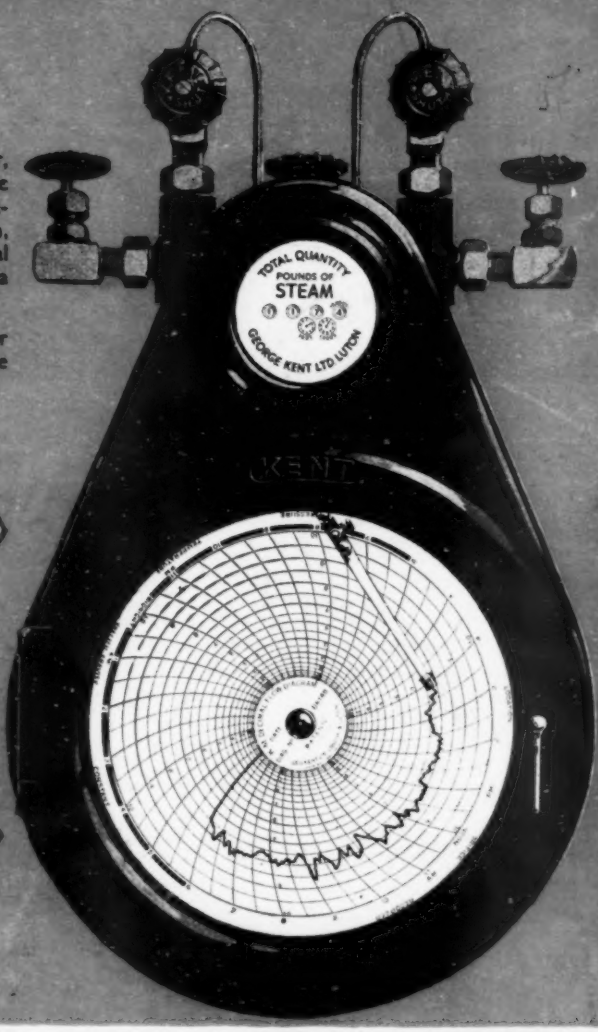

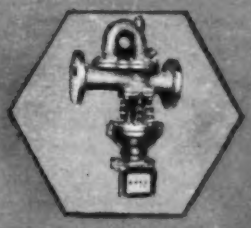

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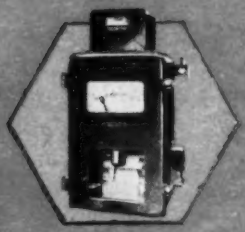

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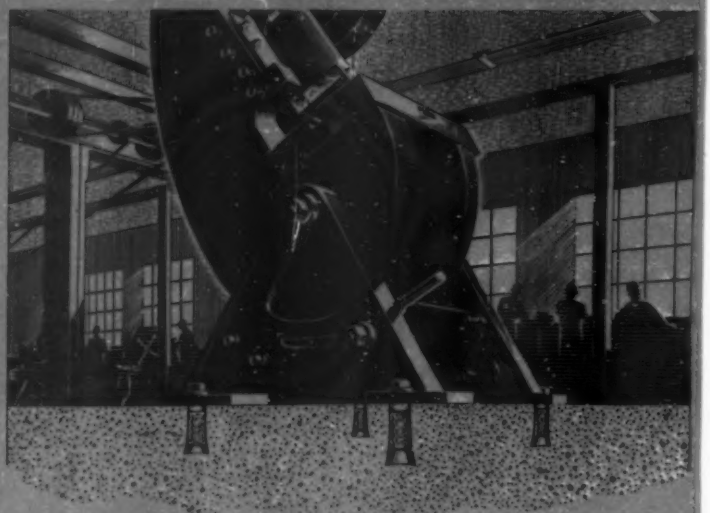

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METALLURGIA

The British Journal of Metals

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METALLURGIA

THE BRITISH JOURNAL OF METALS.
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APRIL, 1943 VOL. XXVII. No. 162

Development of Austenitic Manganese-Alloyed Steels for Aero-Engine Exhaust Valves

By D. W. Rudorff, M.Inst.F.

Since the war began many modifications have been made in the composition of valve steels, and in this review attention is directed particularly to a special investigation recently carried out at the Materials Investigation Department of the German Association for Aircraft, designed to ascertain the relative merits of Austenitic Cr–Mn–Ni and Cr–Ni valve steels.

THE present extended emergency has necessitated a reconsideration of the composition of many special alloy steels. War on the present gigantic scale has led to a demand for the alloying elements far in excess of their availability, and much attention has been directed to possible modifications in compositions so that the properties of a particular type of steel may be retained. For aero-engine exhaust valves, for instance, austenitic steels containing 0·45% C, 1·5% Si, 0·8% Mn, 15% Cr, 13% Ni, and 2·5% W are most frequently used in German practice.[1] Other austenitic valve materials used in certain instances are alloyed within the following limits—0·3–0·6% C, 0·5–2·0% Si, 0·5–1·5% Mn, 8–28% Ni, 12–24% Cr, 1–5% W, with occasional additions of small amounts of Mo or V. Austenitic steels with more than 1·5% Mn have not yet been employed as valve material by the German engine-makers. For this reason a special investigation[2] has recently been carried out at the Materials Investigation Department of the German Association for Aeronautical Engineering, in order to ascertain the relative merits of austenitic Cr–Mn–Ni valve steels as compared to austenitic Cr–Ni steels.

The materials used in this investigation are listed in Table I.

TABLE I.—MATERIALS INVESTIGATED.

Steel No.	C, %	Si, %	Mn, %	Ni, %	Cr, %	W %	V %	Al, %	Specific Weight, Grm./cub. cm.
V 1	0·5	1·73	0·74	13·0	15·8	2·07	—	—	7·951
V 2	0·4	1·5	0·68	10·0	15·8	2·35	—	—	7·939
V 3	0·43	3·0	1·13	8·8	19·2	0·9	—	—	7·782
V 4	0·42	1·51	2·6	6·1	18·2	1·3	—	—	7·846
V 5	0·45	1·68	1·03	4·2	18·7	1·00	—	—	7·792
V 6	0·47	1·61	6·63	4·2	13·1	1·06	—	2·55	7·537
V 7	0·43	4·26	6·80	4·3	13·1	1·05	—	—	7·631
V 8	0·23	2·22	2·95	6·6	20·2	—	0·6	—	7·704
V 9	0·5	2·22	3·42	6·0	16·2	—	1·13	—	7·698
V 10	0·55	2·30	1·32	5·3	17·9	—	0·77	—	7·671
V 11	0·53	1·78	3·42	6·2	16·1	0·70	0·86	—	7·739
A	0·95	1·63	0·74	12·8	15·1	2·45	—	—	—
							Mo	Cu	Ta+Nb
B	0·1	?	0·68	15·2	17·6	—	2·2	1·8	1·96
C	0·1	?	0·72	8·9	17·7	—	—	—	0·94
D	0·1	?	8·86	0·64	18·3	—	—	—	—

Previous to testing, all specimens were held at 800° C. for 1 hour and subsequently cooled in air. Referring to Table I, the Cr–Ni–W steel V 1 was originally taken as a base of comparison. The analysis of this material approximates to that used for the exhaust valves of the B.M.W. aero engine, 132K. Compared to steel V 1, the steel V 2 is seen to have a reduced nickel percentage; while in steel V 3 a decrease in tungsten percentage is accompanied by an increase in chromium content. In steels V 4 and V 5 the nickel contents are lowered to 6·1 and 4·2%, respectively; while their manganese contents are increased to 2·6 and 4%, respectively. In the case of steels V 6 and V 7 the nickel percentage was maintained at that used with steel V 5, but the chromium content was lowered to 13·1%. In consideration of the aluminium content of 2·5% of steel V 6, the manganese content of this material had to be chosen as high as 6·6% in order to maintain its austenitic character. Similarly, the silicon content of 4·26% of steel V 7 necessitated the addition of 6·8% manganese to this material.

While steels V 1—V 7 contain from 0·9—2·35% W, the steels V 8—V 10 are alloyed with 0·6—1·13% V instead. Steel V 8 is remarkable for its low-carbon content of 0·23%. The composition of steel V 11 is similar to that of steels of V 8—V 10, but it contains 0·7% W in addition. The steels A—D were included in the tests only for the sake of clarifying the problem of nitriding, and it is seen that steel A differs from steel V 1 only with regard to the higher-carbon content. The low-carbon steel B contains Mo, Cu, Ta and Nb in addition to Ni and Cr. The Cr–Ni steel C is likewise alloyed with Ta and Nb, while steel D represents a Cr–Mn material with low Ni percentage. Steels A and B possess a purely austenitic ground mass. In contrast to this, steel C contains 10—20% ferrite and steel D shows 30—40% ferrite.

An examination of steels V 1—V 11, after heating at 800° C. for 1 hour, showed that all specimens possessed an austenitic ground mass with carbide inclusions. The absence of any noteworthy ferromagnetism proved that none of the specimens contained ferrite or martensite. After holding at 700° C. for 100 hours, all specimens exhibited austenitic structure with carbide inclusions.

Inspection of steels V 6, V 9 and V 11 revealed that in the specimens subjected to long-time heating at 700° C. a pronounced carbide precipitation on the grain boundaries had taken place. Steel V 6 had become distinctly ferromagnetic, and the magnetic properties of V 5 and V 9 had also slightly increased. In the case of steels V 5 and V 9 this tendency can be suppressed by small increases in Ni and Mn percentage, but much larger percentages would be required with steel V 6.

Referring to the specific weights listed in Table 1, it is seen that steel V 1 and V 2 register the highest values in

1 cf. "Metallurgical Study of German and Italian Aircraft Engine and Airframe Parts." METALLURGIA, October, 1942, p. 208.

2 H. Cornelius. "Properties of Austenitic Aircraft Engine Valve Steels alloyed with Manganese." *Luftfahrtforschung*, vol. 19, 1942, Lfg. 2, p. 44.

this respect, while steel V 6 (containing 2·55% Al) has the lowest specific weight, the extremes being 7·939 in the case of steel V 2 and 7·537 g./cm.³ for steel V 6. The thermal expansion coefficients of the various specimens were determined with the use of a differential-dilatometer, the rate of temperature variation being kept at 1·5° C. per min. According to the data charted in Fig. 1 the mean expansion coefficients of steels V 3—V 11 with small nickel, and correspondingly increased manganese percentage, increase more sharply with the temperature than the coefficients found with steels V 1 and V 2 containing more nickel and less manganese. On the other hand, the latter two steels exhibit a greater thermal expansion coefficient below 400° C. Steel V 6 is shown to possess the lowest thermal expansion coefficient of all specimens investigated, its average value being $18{\cdot}0 \times 10^{-5}$ mm.mm.$^{-1}$.°C. for the range of 20°—800° C.

Measurement of the specific heat conductivity of the various steels yielded the values listed in Table II :—

TABLE II.—MEAN THERMAL CONDUCTIVITY (Cal./cm. sec. ° C.).

Steel.	50°—100° C.	100°—200° C.	200°—300° C.
V 1	0·028	0·029	0·03
V 2	0·0319	0·0298	0·032
V 3	0·0293	0·0289	0·0308
V 4	0·0284	0·0297	0·031
V 5	0·0284	0·030	0·0309
V 6	0·0265	0·0281	0·0289
V 7	0·0275	0·0268	0·0288
V 8	0·0288	0·0285	0·0309
V 9	0·0281	0·0292	0·0304
V 10	0·0281	0·0290	0·0302
V 11	0·0282	0·0293	0·0298

Here it is seen that the values found for the steels with increased manganese percentage are hardly different from those registered for specimens high in nickel and low in manganese. Only steel V 6, which contains aluminium, shows a somewhat inferior heat conductivity as compared to steel V 1.

Wear Properties

For the investigation of the wear resistance, a specially devised apparatus was used in which a spherically rounded-off piece of the test specimen is made to travel through an arc of 22 mm. length, the piece being loaded with a weight of 1 kg., and the average travel speed being 0·733 m./sec. Two series of test runs extended over 2,200 m. and 13,200 m. length of travel, respectively, for each run, no lubricant of any kind being employed. The resultant wear is expressed by the diameter of the worn circular surface. Prior to the test runs the polished surfaces of both specimen and cast-iron plate were slightly roughened. For the cast-iron plate a material with 2·6% C, 2·2% Si, 0·7% Mn, 0·077% P, 0·057% S, 1·4% Ni and 1·15% Mo was used. Heat-treatment of the plate, consisting of oil-quenching from 760° C. followed by a 1-hour draw at 500° C., resulted in a Brinell hardness of the plate surface of 344—353.

The results obtained with this test method were expressed by the diameter of the worn surface. No definite relationship between wear resistance and hardness can be claimed to exist. Nor was it possible to detect any relationship between grainsize and wear resistance. The smallest resistance to wear was exhibited by steel V 8, which has the lowest carbon percentage of all specimens tested. However, the wear of steel V 6 containing the largest carbon percentage was only little less than that of steel V 8. While the steels containing manganese exhibited a somewhat greater wear, the results of this wear test as a whole cannot be considered to furnish any conclusive proof in this direction.

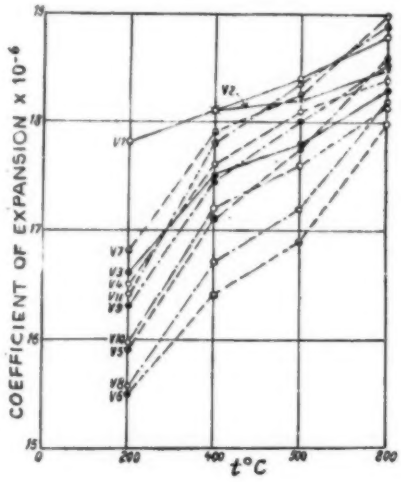


Fig. 1.—Mean thermal expansion coefficients between 20°—800° C.

TABLE III.

Steel No.	Hardness before Nitriding, Brinell.	Core Hardness after Nitriding, Brinell.	Method 1* Surface Hardness, Brinell.	Method 2† Surface Hardness, Brinell.	Method 3‡ Surface Hardness, Brinell.	Method 4† Hardness Surface, Brinell.	Method 4† Hardness 0·03 mm. below Surface, Brinell.	Method 4† Depth of Nitrided Layer, mm.	Method 5†‡ Hardness Surface, Brinell.	Method 5†‡ Hardness 0·03 mm. below Surface, Brinell.	Method 5†‡ Depth of Nitrided Layer, mm.	Method 6†§ Hardness Surface, Brinell.	Method 6†§ Hardness 0·03 mm. below Surface, Brinell.	Method 6†§ Depth of Nitrided Layer, mm.
V 1	200	206	256	250	254–945	834–895	700	0·105–0·11	1060–1150	1020	0·109	1050–1065	824	0·10–0·11
V 2	212	210	215	542	230–815	752–803	910	0·155–0·16	860–1130	1245	0·163	770–895	1033	0·17
V 3	235	224	237	245	260	235	—	0	240	—	0	680–745	1170	0·14–0·16
V 4	238	237	230	245	300	263	—	0	252	—	0	260–750	—	0·128
V 5	250	260	293	260	297–383	270	—	0	275	—	0	260–1130	1100	0·0–0·12
V 6	220–300	235	300	290	421–642	743–762	895	0·175–0·195	256	—	0	810–1010	824	0·26
V 7	260	257	270	250	322–383	315	—	0	320	—	0	350–880	1205	0·0–0·15
V 8	225	225	232	295	240	245	—	0	225	—	0	1080–1100	1200	0·15–0·16
V 9	270	270	300	290	290–385	317	—	0	270	—	0	365–824	1065	0·0–0·14
V 10	270	272	310	305	300	290	—	0	225	—	0	300–700	—	0·0–0·16
V 11	270	274	260	275	290–317	322	—	0	280	—	0	650–800	1020	0·20
A	250	242	365	450	297–824	660–720	790	0·09–0·1	820–824	1000	0·091	860–930	790	0·08–0·09
B	150	178	180	210	214	1000–1030	810	0·44–0·145	830–975	1130	0·132	900–1020	1290	0·142
C	160	153	190	193	206–724	1030–1260	810	0·17–0·195	850–950	1000	0·205	840–900	1130	0·23–0·28
D	200	208	205	212	210	195	—	0	210	—	0	210–770	510	0·0–0·16

* Length of nitriding treatment, 100 hours.

† Length of nitriding treatment, 90 hours.

‡ Data regarding surface hardness and depth of layer refer to polished and ground specimens.

§ Data given for steels V 1, V 2, V 3, V 6, V 8, V 11, A, B and C refer to ground and polished specimens. Data given for the other steels refer to ground and polished specimens (small surface hardness and nitriding depth), and also to dressed specimens (high surface hardness and great nitrided layer depth).

Nitriding Properties

The nitriding of steels with high-chromium content, as also that of austenitic chromium-nickel steels, is made difficult by the presence of oxidising agents in the metal surface, which tend to form a thin oxide layer. All specimens to be nitrided were quenched from 1,050°—1,100° C. previous to the test. Especial attention was paid to the preparation of the surface, six different procedures being resorted to.

In the first method the metal surface was covered with vaseline in order to prevent oxidation, and then ground with emery paper No. 0. Degreasing was carried out immediately before inserting the specimen in the nitriding furnace. The second method differed from the first only in so far as emery paper No. 000 was used. In the third method the specimen was polished prior to applying method No. 2, while the fourth method consisted in applying emery paper No. 0, followed by degreasing and subsequent electrolytic copper-plating to a film thickness of 0·15—0·25 microns. Method No. 5 was similar to No. 4, except that the pieces were pickled in 10% sulphuric acid prior to copper-plating. In the sixth method the cadmium - phosphate treatment[3] employed by Jones was followed.

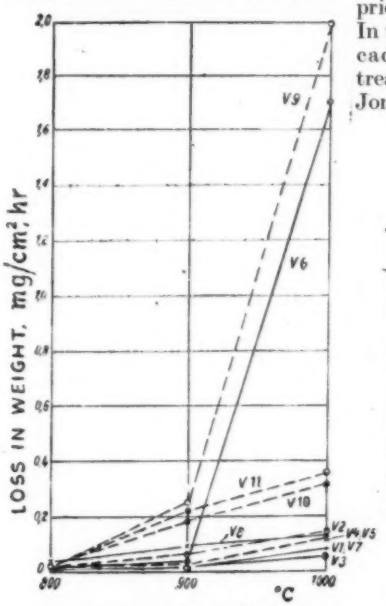


Fig. 2.—Loss in weight due to scaling in air at various temperatures.

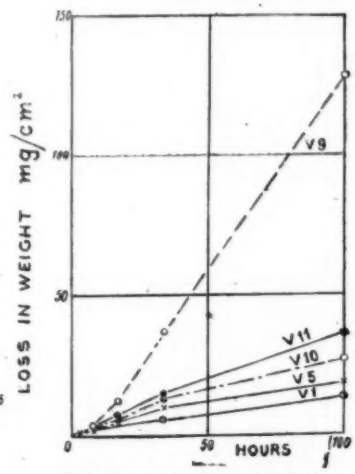


Fig. 3.—Loss in weight due to scaling at 1,000° C. versus time.

The results of the nitriding tests are tabulated in Table III, which shows that the difficulties encountered in the nitriding of the manganese alloyed steels can be overcome with the use of the aforementioned method No. 6. A remarkable fact, yet unexplained, is that successful nitriding of the manganese steels is conditional upon the employment of dressed surfaces, ground or polished surfaces giving entirely unsatisfactory results. With regard to the Cr-Ni steel specimens, methods Nos. 1—3 are seen to be unsatisfactory. While methods Nos. 4—6 gave generally good results, the methods 5 and 6 are particularly recommended; but it should be noted that in the case of the Cr–Ni steel V 3 only method No. 6 is effective.

Besides evidencing that nitriding of manganese-alloyed steels may be impossible under conditions which lead to the successful nitriding of Cr–Ni steels, Table III also furnishes valuable information in other respects. First of all, the results obtained for steels V 1, V 2 and A with method No. 6 prove the well-known fact that with austenitic steels the thickness of the nitrided layer decreases with increasing nickel content.[3] Furthermore, it can be deduced that the carbon percentage is only of insignificant influence upon the thickness of the nitrided layer, the layer produced in steel A containing 0·95% C being little less than that observed with steel V 1 containing 0·5%. However, as shown by the results obtained with steel A, the hardness of the nitrided layer appears to decrease in the case of high-carbon Cr–Ni materials.

Concerning steel V 3, the small excess in manganese content over the usually employed percentage is seen to prove a considerable obstacle to satisfactory nitriding, only method No. 6 proving effective. The fact that the manganese alloyed material V 4—V 11 exhibit a greater depth of the nitrided layer leads to the conclusion that, once conditions for successful diffusion of the nitrogen into the material are established, the inhibiting effect of manganese is smaller than that of the nickel. Thus, steels A and V 1 having the highest nickel contents next to steel B show smallest depth of the nitrided layer. Finally, it must be noted that steel B with 1·8% Cu could not be successfully nitrided in contrast to austenitic low-alloyed steels containing 1·2% Cu.[4]

Generally, greatest hardness of the nitrided layer is encountered approximately 0·03 mm. below the surface, as shown by the results given in Table III. But in cases where high hardness is recorded for the very surface of the nitrided layer, a decrease in hardness immediately below the surface can generally be expected.

Scaling Resistance

Scaling resistance tests were conducted in still air at 800°, 900° and 1,000° C., the test duration being 96 hours. Every 24 hours the test-pieces (of 10 mm. dia. and 25 mm. length) were removed from the furnace, cooled to room temperature, and then put back into the furnace. According to the chart reproduced in Fig. 2, up to 800° C., all specimens possess a high degree of resistance to scaling. At 900° C. increased scaling takes place with steels V 9, V 10 and V 11, while steel V 6 exhibits poor scaling resistance at 1,000° C. At the latter temperature, particularly poor results were obtained with steel V 9. The general conclusion can be drawn that partial replacement of nickel by manganese is without adverse influence upon the scaling properties of the materials concerned. In view of the experiences reported by Cornelius and Bungardt, it may well be conceived that the deterioration in the scaling resistance of steels V 9 and V 11 is due to the presence of more than 0·6% V in combination with a high-nickel percentage. This appears to be confirmed by the fact that taking the steels in the order of their V content, that is V 10 (0·77% V), V 11 (0·86% V), and V 11 (1·13% V), their scaling properties are seen to deteriorate in the same order.

TABLE IV.—MECHANICAL STRENGTH AFTER EXPOSURE TO ELEVATED TEMPERATURE.

Steel No.	0·2% Yield Point, Lb./sq. in		Ultimate Tensile Strength, Lb./sq. in.		Elongation, %.		Impact Strength, Ft./lb.	
	800° C., 1 hr.	700° C., 100 hrs.	800° C., 1 hr.	700° C., 100 hrs.	800° C., 1 hr.	700° C., 100 hrs.	800° C., 1 hr.	700° 100 hrs.,
V 1	45,140	41,175	110,760	108,630	34·3	33·5	47·7	44·5
V 2	42,600	52,540	117,860	127,800	36·5	31·7	40·5	41·9
V 3	49,700	45,440	137,200	127,800	37·5	36·7	43·0	44·5
V 4	65,320	—	151,940	140,580	24·3	23·0	31·1	30·6
V 5	45,400	65,320	150,520	163,300	22·0	27·0	29·6	15·2
V 6	69,580	77,390	116,440	151,940	13·0	2·0	15·2	1·4
V 7	48,990	43,310	159,040	168,950	18·3	15·0	14·9	13·0
V 8	48,275	46,860	115,020	112,890	14·2	36·0	33·8	30·1
V 9	73,125	63,900	164,720	157,620	27·6	22·5	31·1	31·6
V 10	56,800	52,540	139,040	154,780	26·5	21·8	31·8	27·7
V 11	58,930	53,250	156,200	139,160	26·0	16·1	27·7	26·8

3 B. Jones. "Further Experiments on the Nitrogen Hardening of High Cr and Austenitic Steels," *Jl. Iron & Steel Inst.*, vol. 136, 1937, p. 169.

4 Sun-Ichi-Satoh. "Influence of Nitrogen on Special Steels and some Experiences on Case-hardening with Nitrogen," *Trans. Am. Inst. Min. & Met. Engrs., Iron and Steel Div.*, vol. 90, 1930, p. 192.

Fig. 3 shows the scaling properties of steels V 1, V 5, V 9, V 10, and V 11 at 1,000° C. as a function of the time of exposure; and it is clearly seen that steel V 9, possessing the highest V percentage, shows greatly increased scaling with extended periods of exposure.

Tensile and Impact Strength Tests

Tensile and impact strength tests at room temperature were preceded by holding the specimens at 700° C. for 1 hour and at 700° C. for 100 hours, respectively. The specimens used for the tensile strength test had a diameter of 10 mm., and a measuring length of 50 mm. For the notched-bar test, pieces of 10 × 10 × 55 mm., with a rounded notch of 4 mm. depth, were used. Referring to the test data tabulated in Table IV, it is seen that compared to the results obtained with 1-hour treating at 800° C., the 0·2% yield point, as well as tensile strength and impact strength of the Cr–Ni–W steels V 1, V 2 and V 3, are hardly affected by heating for 100 hours at 700° C. Generally, the tensile strength of these steels is seen to be smaller than that of the manganese-alloyed materials, steel V 6 (after heating at 800° C.) and steel V 8 (after heating at both 800° and 700° C.) being the sole exceptions.

While 100-hour heating at 700° C. is of little influence upon the mechanical properties of the Cr–Ni–Mn–W steel V 4, containing 2·60% Mn, steel V 5, with 4·03% Mn, undergoes important changes in its mechanical properties. Its ultimate tensile strength and its 0·2% yield point are increased, its elongation is moderately decreased, and its impact strength falls sharply. These changes must be considered due to partial decomposition of the austenite into martensite. Even more pronounced changes of this nature occur with steel V 6, and it is quite evident that the manganese and/or nickel content of these two steels is insufficient to ensure complete austenite stability during long-time heating at 700° C. Under the same conditions steel V 7 retains good austenite stability; its tensile strength increases little, and elongation and impact strength register small decreases. The same holds true of the steels V 9, V 10 and V 11. It can, therefore, be concluded that manganese steels can well be made to approximate to the properties of steel V 1.

Retention of Hardness at Elevated Temperature

For the determination of the interrelationship between hardness and temperature, a specially constructed pendulum type instrument was used in which a small furnace is incorporated, so as to hold the specimen at the desired temperature during the hardness test. Referring to Fig. 4, it will be seen that the divergence of hardness values registered with the various materials considerably decreases with the temperature. Compared to a divergence of *ca.* 125 Brinell units at room temperature, this range has decreased to 40 Brinell units at 400° C., and to 30 units at 900° C. The graph also shows that generally the hardness of the manganese-alloyed steels approaches that of the Cr–Ni steels, the only exception being the low-carbon Cr–Ni–Mn–V steel V 8, which shows the lowest of all hardness values recorded.

Resistance to Creep at Elevated Temperature

The creep-strength tests were carried out at 700° C., after the specimens had been held at 800° C. for 1 hour. In each case the creep test extended over 300—360 hours, and creep-strength values were established on six different bases, viz.:—

(1) Stress s_1 corresponding to $10 \cdot 10^{-4}$% elongation per hour during the 25—35 test hours, the permanent deformation after 45 hours being less than 0·2%.

(2) Stress s_2 corresponding to $5 \cdot 10^{-4}$% elongation per hour during the 25—35 test hours, the permanent strain after 45 hours being less than 0·2%.

(3) Stress s_3 corresponding to a strain of $5 \cdot 10^{-4}$% per hour during the 100—300 hours of the test.

TABLE V.—CREEP TEST DATA.

Steel No.	Tensile Stress corresponding to Creep Rate per hour of $10 \cdot 10^{-4}$% during 25—35 hours. s_1° Lb./sq. in.	$5 \cdot 10^{-4}$% during 25—35 hours. s_2° Lb./sq. in.	$5 \cdot 10^{-4}$% during 100—300 hours. s_3° Lb./sq. in.	$1 \cdot 10^{-4}$% during 100—300 hours. s_4 Lb./sq. in.	Stress causing 1% permanent deformation in 300 hours. s_5 Lb./sq. in.	Creep Rate produced by Stress s_4 during 25—35 hrs. r_4 10^{-4}% per h.
V 1	5,396	4,828	3,976	3,408	6,106	—
V 2	10,650	8,804	9,372	6,816	—	2·8
V 3	7,242	7,100	7,242	6,674	7,952	1·2
V 4	8,236	7,526	8,236	5,822	—	1·0
V 5	9,656	7,810	9,372	6,958	—	4·0
V 6	1,958	1,136	—	—	3 129	—
V 7	8,804	6,390	8,804	6,816	—	5·3
V 8	7,952	4,680	6,958	4,396	8,520	5 6
V 9	9,656	8,804	9,514	7,952	—	2·5
V 10	9,940	6,390	8,946	5,680	—	4 7
V 11	8,236	6,816	7,810	5,680	—	2 5

[a] Permanent elongation after 45 hours always smaller than 0·2%.

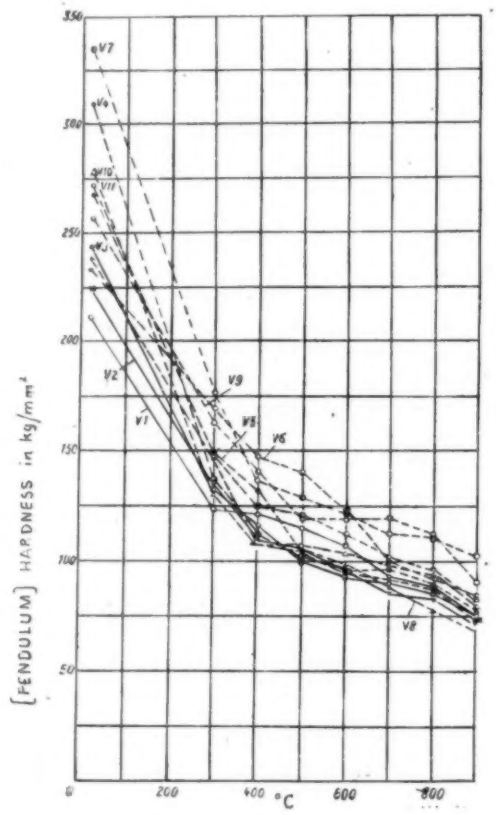


Fig. 4.—Hardness versus temperature.

(4) Stress s_4 corresponding to a strain of $1 \cdot 10^{-4}$% per hour during the 100—300 hours of the test. This corresponds to a creep of 1% in 10,000 hours, provided the creep rate remains constant after the 300 hour.

(5) Stress s_5 corresponding to a creep of 1% in 300 hours. This value is applicable to parts which are not expected to have a service life greater than 300 hours.

(6) Creep rate produced during the 25—35 test hours by the stress s_4.

Table V shows that the stresses found with method 1 are consistently above those obtained with method 4; while the stresses found with method 3 are either equal to, or smaller than, those found with method 1. It must, therefore, be concluded that the stresses s_1 will produce a creep rate of at least $5 \cdot 10^{-4}$% per hour during the 100—300 hours. This points to an increase in the creep rate during long-time loading with stress s_1. With the exception of cases in which service life is limited to 300 hours, creep strength, therefore, should be determined on the basis of method 4.

In view of the unusually low-creep stress s_4 registered with steel V 1, this steel must be omitted in assessing

the comparative creep properties of Cr–Ni steels *versus* Cr–Ni–Mn steels. Referring to Table V, the stresses s_4 found for the Cr–Ni–W steels V 2 and V 3 are seen to be equalled or exceeded by the stresses s_4 found for the Cr–Ni–Mn–W material V 5, and for the Cr–Mn–Ni–Si–W steel V 7, as well as for the Cr–Ni–Mn–V steel V 9; while the manganese-alloyed steels V 4, V 10, V 11, V 8 and V 6 show progressively declining creep strength s_4, which is inferior to that of steels V 2 and V 3 Particularly poor creep strength at 700° C. is displayed by the aluminium-alloyed material V 6. Considering the low-carbon content of the Cr–Ni–Mn–V steel V 8, its creep strength is highly satisfactory. In this connection it is interesting to note that according to A. M. Borsdyka and K. A. Lanskaya,[5] variations in carbon content within a range of 0·15—0·45% are of little influence upon the creep properties of 14% Cr–14% Ni–2·5% W steels.

The creep-strength values found with steels V 8—V 11 may be taken as an indication to the effect that tungsten can be safely replaced by vanadium without impairing the creep strength of these valve steels. But in view of the adverse influence of vanadium upon the scaling properties, vanadium contents higher than 0·5—0·6% should not be used. Judging from the satisfactory characteristics displayed by steels V 4 and V 5, the German investigator comes to the conclusion that a material containing 0·42—0·52% C, 1·5—2·0% Si, 3·0—5·0% Mn, 6·0—4·5% Ni, 17·5—18·5% Cr and 0·9—1·3% W should prove a very satisfactory valve material.

5 A. M. Borsdyka and K. A. Lanskaya. *Metallurgh*, 1940, No. 10, p. 25.

Desulphurising of Cast Iron

The practical use of soda ash and caustic soda for refining and desulphurising cupola iron is considered, and some results obtained are given. Under favourable conditions upwards of 85% of the sulphur in cupola irons, within the range from 0·06 to 0·13% sulphur, can be removed with caustic soda in one treatment without external heat.

THE use of alkalies for refining and desulphurising cupola iron in foundries has developed during recent years along three separate and distinct lines—namely (1) desulphurising and refining in mixer ladles as a means of improving the physical properties of metal; (2) desulphurising and refining in the cupola with improvements in melting operations; and (3) desulphurising in duplexing processes where the principal object is the removal of sulphur. Refining and desulphurising of cast iron with soda ash or caustic soda[1] is being increasingly applied, and in a recent article by G. S. Evans[2] the practical use of these reagents as ladle or cupola additions is dealt with and some results obtained are described.

In desulphurising and refining in mixer ladles, the best results are obtained when the soda ash is limited to from 2 lb. to 15 lb. per ton, and the metal held in the ladle in contact with the soda slag from 3 to 20 mins., with sulphur reduction of from 10 to 50% varying with the mixture and class of casting produced. In refining metal for pouring light castings, sulphur reduction is generally limited to 20 to 30% of the sulphur content of the iron at the cupola spout in order to prevent loss of temperature. With metal for heavy castings which can be poured at a lower temperature, sulphur reduction of 40 to 50% or more is quite satisfactory. The maximum benefit of the refining as expressed by the effect upon the casting of the metal generally coincides with the removal of occluded gases expelled by the boiling action set up by the treatment and the removal of oxide slags and other non-metallic impurities which are washed out of the iron and carried off in the soda slag. The development of a U-shaped refining ladle with cover and insulation has extended refining in many production foundries, and although not adapted to maximum per cent., desulphurisation is particular efficient in providing a continuous supply of refined metal.

There has been a gradual increase in the use of fused soda ash in the cupola during the last few years, since it has been shown that with its proper application in the cupola, particularly when melting high steel mixtures which are notably hard on the lining, soda ash serves to save the lining as well as to save the oxidation of the metal. Added in the cupola along with limestone, soda ash serves the dual purpose of improving melting conditions by making a more fluid slag, by accelerating the combustion reactions, and by preventing some sulphur pick-up by the iron. In average cupola practice, 4 lb. of soda ash per ton of metal added along with the limestone charge will lower the sulphur in the iron at the spout by about 15%. With high-steel mixtures which tend to pick up more sulphur during melting the effect is nominally more pronounced, while with straight cast scrap and pig-iron charges the effect on sulphur pick-up is probably less.

With batch or individual ladle treatment in duplexing processes, such as cupola-converter or cupola-electric steel operations, where the prime object of desulphurising is maximum sulphur reduction, and provided cupola melting practice is good, the sulphur in iron from the cupola can be reduced by 60 or 70% with soda ash, and upwards of 85% with caustic soda without lowering the temperature of the iron below the blowing range.

The most important factors in efficient desulphurising, excepting temperature of the metal, are clean metal, free of slag or any silicious impurities, and rapid and intimate contact between the metal and the soda slag. In order to ensure clean iron, it is essential to use plenty of limestone in the cupola and, if necessary, a slag liquefier to produce a good fluid slag which will rise out of the iron in the hearth of the cupola and always stop off the tap before any slag shows on the stream of iron in the spout. As soda slag is neutralised by any cupola slag or silicious materials present, it is necessary that the ladle should be free of loose dirt or slag before adding the reagent, and every precaution should be taken to prevent any cupola slag coming out with the metal, otherwise the soda slag is diluted and its desulphurising value dissipated.

To ensure both clean iron and maximum contact between the soda slag and the metal, the cupola tap-hole should be smaller than the average, so as to extend the tapping time over a longer period, thus prolonging the stirring action by the stream of iron, and a small head of iron should always be held back in the cupola at the end of the tap, so as to prevent any cupola slag flowing out with the last of the iron. The stream of iron flowing into the desulphurising ladle promotes slag metal contact, both due to the physical effect of the stream in stirring the metal, and also to its action in whipping portions of the freshly melted or undecomposed alkali down into the metal bath. This not only increases the extent of the slag metal contact, but also makes for more effective contact because of the pressure of the overlying metal. The decomposition of the

1 METALLURGIA, 1939, vol. 20, pp. 153–154.
2 *Metals and Alloys*, 1942, vol. 16, No. 5, pp. 876–880.

reagent under the metal surface with the consequent evolution of gases also causes a violent boiling action which helps to increase surface contact still further.

When desulphurising in 1 to 3-ton capacity ladles, with two cupola taps required to fill the ladle, and when using fused soda ash in 2 lb. pigs, about two-thirds of the reagent should be placed in the bottom of the ladle before the first tap and the balance just after stopping-off the cupola. The first charge gradually melts, providing a good volume of a constantly active slag, which is continuously whipped into the bath during the first tap. In the second addition the pigs penetrate the slag and melt in contact with the metal, gradually releasing fresh or undecomposed alkali which causes continued boiling action and makes for maximum slag metal surface contact throughout the tap. If the ladle is filled in one continuous tap, one half of the reagent is added at the start and the balance during the middle third of the tap. The time required for the completion of the reaction, which is indicated in part by quieting down of the slag, will usually vary from 5 to 10 mins., after the last tap in 1 to 3-ton ladles, to upwards of 15 to 20 mins. in larger capacity ladles, where maximum desulphurisation is required as in converter steel practice.

The alkalies available for use in commercial desulphurising practice include, in the order of their activity, (1) Caustic soda; (2) soda ash-caustic soda mixtures; and (3) soda ash. The active agent in each is sodium oxide, of which soda ash contains 58% and caustic soda 76%. Under favourable conditions upwards of 85% of the sulphur in cupola irons, within the range from 0·06 to 0·13% sulphur, can be removed with caustic soda in one treatment without external heat. The maximum reduction obtainable when treating similar irons with soda ash is about 75%.

Repeated Stress-Relieving Treatments on Steel Plate

An investigation is described designed to determine how often a given stress-relieving treatment for a given steel can be repeated without lowering the normal temperature mechanical properties of the material involved below its minimum specification values. Results are briefly considered.

IN heavy structural fabrication, it is often necessary to stress relieve by annealing a part or unit more than once between the beginning of fabrication and its completion. The question has been raised as to how often a given stress-relieving treatment for a given material can be repeated without lowering the normal temperature mechanical properties of the material involved below its minimum specification values. To determine this point, an investigation has been carried out recently by F. Eberlea and T. Brick* on different types of steel.

The steels tested included a 27 to 31 tons per sq. in. 1 in. thick steel plate, containing 0·25% carbon and 0·57% manganese; a 31 to 35 tons per sq. in. 4 in. thick steel plate, containing 0·31% carbon and 0·77% manganese; a molybdenum steel plate, 1 in. thick, containing 0·21% carbon and 0·56% molybdenum; a molybdenum steel plate, 4 in. thick, containing 0·15% carbon and 0·48% molybdenum; and a manganese-vanadium steel plate, $\frac{3}{16}$ in. thick, containing 0·16% carbon, 1·24% manganese, and 0·10% vanadium. Each of the steels was subjected to 10 stress-relieving treatments at the temperatures employed in actual fabrication—namely, 600° C. for the carbon steels and 650° C. for the alloy steels. The initial or standard treatment consisted in holding the material at the prescribed temperature for 1 hour per inch of thickness. This cycle was repeated successively up to 10 times for each material and different plate thickness. The tensile and impact properties of each steel were then determined on four test specimens.

The yield-points of the 1 in. carbon steel plate, the 4 in. carbon steel plate, and the 4 in. molybdenum steel plate remained practically unaffected up to 10 stress-relieving treatments. The yield-points of the 1 in. molybdenum steel plate and the manganese-vanadium steel plate decreased by approximately 15 and 7% respectively during the first seven stress-relieving treatments, after which they remained practically constant. They did not drop below the specified minimum value. The initial yield-point of the 4 in. carbon steel plate was very close to the specified minimum value, and in two instances fell slightly below the latter. The tensile strength of all the materials showed a decrease with increased stress-relieving time, the decrease being lowest for the carbon steel plates. The tensile strength of the 4 in. carbon steel plate and the two molybdenum steel plates fell below the specified minimum value after three to five stress-relieving cycles.

The elongations of all the steels increased slightly with increased stress-relieving time, that of the 1 in. molybdenum steel plate being more pronounced than with the other materials. The reduction of areas of the carbon steel plates increased only slightly with increased stress-relieving time, while the increase of those of the other three materials was more marked. Charpy impact values of the 1 in. carbon steel plate remained practically constant after a slight initial drop, those of the 4 in. carbon steel plate showed a slight uniform decrease, and those of the other three steels a slight uniform increase.

The microstructures of all the steels showed progressive spheroidisation of carbides with increasing stress-relieving time. The two molybdenum steels appeared to display a greater resistance to carbide spheroidisation than the three other materials. The manganese vanadium steel appeared to possess the highest rate of spheroidisation. In general, the results indicated that up to about four stress-relieving treatments the spheroidisation of the carbides had not progressed far enough so as to seriously impair the creep resistance of the materials. This held true of the 1 in. thick material as well as of the 4 in. thick material.

The results obtained show that stress-relieving treatments may be repeated up to 10 times without danger of appreciably lowering the normal temperature mechanical properties of the materials tested below the minimum values specified. The temperatures involved produce progressive spheroidisation of carbides. After about four to five stress-relieving treatments coalescence of carbides, which is known to lower the resistance to creep under certain conditions, becomes noticeable. In the light of the available knowledge on this subject and on the basis of the experimental results obtained, it appears that about five stress-relieving treatments may be applied to a structure without appreciably lowering the normal mechanical properties or the creep resistance of the materials involved. It is also advisable to keep the temperature of stress-relieving as low as possible. This applies particularly to intermediate stress-relieving treatments.

* *Metals and Alloys*, 1942, vol. 16, No. 5, pp. 894-901.

METALLURGIA
THE BRITISH JOURNAL OF METALS.
INCORPORATING "THE METALLURGICAL ENGINEER"

Co-operative Research in the Metal Industries

THE gulf that formerly existed between scientific knowledge and its application to the needs of mankind has gradually narrowed until to-day it is generally appreciated that progress is dependent upon the ability to apply scientific knowledge effectively. This increased knowledge has meant increased power control, which has not been used to the full, because our new knowledge has not been properly applied. Under the present revolutions tremendous changes are taking place in which science, with its applications and its consequences, has for the first time converted a world where shortage has been inevitable into one where plenty could be possible; a world of overwork into one that could be a world with adequate leisure, and a world of hostile competing units into one that could be unified and peaceful. To make full use of the possibilities that science offers, however, will need a new driving force — a new incentive to co-operation, which will probably be approached by stages, but scientific research must proceed at a greater rate to achieve the progress needed.

In industry the desire for better and cheaper products has resulted from research, but striking as is the progress that has been achieved, it could be much more rapid if more research workers were available. Before the war the United States, in proportion to national income, spent on scientific research five or six times, and Russia about eight times, as much as Britain. The war has seen an improvement, but we must make up our minds that 1 per cent. (instead of one-tenth of 1 per cent., as before the war) should be our minimum investment in scientific research, and would bring in an enormous return.

The importance with which research is viewed in the United States is indicated by a national report,* a book of 250,000 words, published recently, giving a comprehensive and up-to-date review of industrial research in that country up to within a few months of its entry into the war. The first three and the last of the sixteen "findings" of this report were used by Dr. H. Moore, C.B.E., as an introduction to an address on the above subject he delivered recently on the occasion of the presentation to him of the Institute of Metals Platinum Medal for 1943. The "findings" he quoted are :—

"Continuous and increasing application of science by industry is contributing most significantly to the high standard of American living. Viewed in this light industrial research is a major national resource."

"The United States has become the acknowledged leader in industrial research."

"American industry employs over 70,000 research workers in over 2,200 laboratories at an estimated annual cost, based on average figures, of the order of $300,000,000."

"Industrial research has an ever-widening field, and shows no tendency to terminate or even to be restricted for lack of new opportunity."

In his address Dr. Moore stated that the increasing emphasis now placed on the national and the social aspect is evident in the view that the ultimate justification for industrial research is its effect in raising the standard of living (not necessarily only in the country in which the work is done), and enabling mankind to make better use of natural resources. The use of industrial research for restrictive purposes cannot be defended, but as a powerful aid in producing wealth it has its part to play in furthering the aims of the Atlantic Charter.

Industrial research may be carried on in many different ways. Outside the Soviet Union the bulk of industrial research is undertaken by individual companies and corporations, for their own advantage, and this is particularly true in America. Co-operative industrial research, although a comparatively small proportion of the American total, is important. Bodies such as the American Society for Testing Materials and the American Welding Society form committees which plan research programmes carried out in University, Government, or industrial laboratories, the results being published for the benefit of the world. A development more comparable with the Research Associations of this country is the conduct of research by American Trade Associations. Thirty-five out of 115 of these Associations reporting through one channel are stated* to conduct research activities, but expenditure on these averages less than £10,000 per annum per Association. There is no Government contribution.

There are other significant forms of American co-operative industrial research, but the authors of this report considered Germany and Great Britain to be further advanced in co-operation. For example, "research for the benefit of the entire industry," is regarded as "the weakest phase of ferrous metallurgical research in the United States," although the industry is reported to spend at least £2,000,000 per annum on research.

Germany, once a leader in beneficent applied research, has in the past ten years not only demonstrated on the largest scale how easily industrial research can be perverted to the worst purposes, but has struck deeply at the intellectual foundations of science. No doubt this will recoil on Germany by withering the roots of scientific progress there, but only a long time after Germany's vast scientific resources have been used with such terribly destructive effect.

Dr. Moore states that nothing comparable with this American review has been attempted in this country, but the results of the detailed survey of British industrial research, now being made by the Federation of British Industries, is being looked forward to with great interest.

As during the war of 1914–1918, much attention is being directed to post-war reconstruction. It is noteworthy that one piece of reconstruction which survived reaction and the economy axes after the last great war war the research association movement. It is sometimes thought that this scheme arose at, or towards, the end of that war, but the early steps were taken in its first year when the Advisory Council of the body, which soon afterwards became the Department of Scientific and Industrial Research was set up.[1] In May, 1915, the Government

The fact that goods made of raw materials in short supply owing to war conditions are advertised in "Metallurgia" should not be taken as an indication that they are necessarily available for export.

had already decided to establish machinery empowered to promote scientific research for industrial purposes. The "Imperial Trust for the Encouragement of Scientific and Industrial Research" was set up in 1916, and by June, 1917, the co-operative research association had been worked out in full detail and the famous million pounds provided for its operation. The authors of "The Government Scheme for Industrial Research" (June, 1917[1]) were far-seeing men, and their plan has stood the test of time. Progress since that time has been continuous, and particular attention is directed to the various Research Associations in the metal industries that are actively engaged on co-operative research.

Whatever type of industrial organisation may prevail after the present war Dr. Moore considers a greater degree of co-operation within each large industry, in scientific research and in applying scientific knowledge, to be highly probable, and indeed necessary for promoting the efficiency of the whole industry without duplication and waste of effort. The Research Association is the obvious means for promoting this extended co-operation, not by replacing the research organisations of individual manufacturing concerns, but by undertaking work on large problems common to the industry, and by co-ordinating research carried out in the laboratories and works of members.

This war is teaching us some important things about getting the best return out of science. In the Services research has become operational—that is, scientists go out into the field. They are sent to the battle-fronts to find out the detailed needs of the men on the job, to study how the weapons and devices that have been produced as a result of research stand up to actual fighting. We shall need the same thing in peace time. Instead of the scientist or research worker sticking to his laboratory, he must go out into the field and study the needs of consumers, and this would facilitate co-operative research.

[*] United States National Resources Planning Board. "Research: A National Resource." I.—Relation of the Federal Government to Research. 1938, 255 pp. II.—Industrial Research. 1941, 370 pp.

[1] Reports of the Committee of the Privy Council for Scientific and Industrial Research for 1915–16, 1916–17. See also succeeding Annual Reports (Department of Scientific and Industrial Research).

Forthcoming Meetings

Institute of Metals

The 1943 May Lecture, the thirty-third of the series, will be given by Professor G. P. Thomson, D.Sc., F.R.S., on Wednesday, May 19, at the Institution of Mechanical Engineers, Storey's Gate, Westminster, S.W. 1. The Council is grateful to the Council of the Institution for again placing its hall and rooms at the disposal of the Institute for the purpose of holding this meeting. The subject of the lecture will be "Electron Diffraction." It is of interest to note that the lecturer's father, the late Sir J. J. Thomson, O.M., F.R.S., who was an honorary member of the Institute, was also a May lecturer, his discourse—delivered in 1915—being on the "Conduction of Electricity through Metals."

The London Section has arranged a series of evening meetings to be held at the Junior Institution of Engineers during the summer months. The first was the recent annual general meeting, at which the chairman, Mr. A. J. Murphy, gave an address. The next meeting, on May 10, will be a joint meeting with the local section of the Society of Chemical Industry, at which Mr. E. A. G. Liddiard, M.A., Research Manager of the British Non-Ferrous Metals Research Association, will speak on "Modern Methods of Metallurgical Analysis."

Sheffield Metallurgical Association

A lecture on "Isothermal Transformations," on April 20 at 6-30 p.m., by Mr. T. F. Russell.

On May 4, at 6-30 p.m., Dr. T. Swinden will give a paper on "Organisation for Research and Development."

Reducing Waste in the Use of Fuel

Before the war consumers generally were extravagant in the use of fuel and industry was no exception. The fact that this is an important coal-producing country undoubtedly was responsible for much carelessness in the use of this essential mineral, resulting in considerable waste. It is not easy to alter habits and practices accumulated over a long period, but the National Campaign to save fuel, or rather to reduce waste, is doing much to change the attitude of consumers and in effecting a general fuel care conscientiousness which should be maintained in less difficult times. Changes are being brought about by rationing supplies, by regulation, by scientific investigation, and by what might be called "peaceful persuasion."

It is noteworthy that this latter method is being followed with considerable success by the Sheffield and District Industrial Fuel Economy Committee. This Committee was formed under the ægis of Mr. H. S. Houldsworth, K.C., Regional Controller of the North Eastern Region, by the Sheffield Chamber of Commerce, and comprises representatives of local industry, the technical colleges, and those responsible for the actual supply of the very large amount of coal that is required. The Committee, under the chairmanship of Mr. Colin Laycock, meets regularly to report progress and to decide further action in the coal campaign. The system adopted has been to leave the conduct of the technical side to a small panel of experienced fuel technologists, under the chairmanship of Mr. H. C. Armstrong, M.Inst.C.E., who are accustomed to meet daily all kinds of fuel problems. With the help of other fuel engineers loaned by local firms, works are visited and advice given, followed by a written report from the Technical Panel.

A staff of experts is available to help at any time, without introducing vexatious regulations or suggesting large capital expenditure. As might be expected, many interesting facts are brought to light, but surprising has been the number of cases where coal is accepted as a necessary aid to production without any close quantitative analysis being kept; it has, of course, been shown, without doubt, how conducive to saving is the correct measurement of all coal burnt. It is natural that occasionally an adviser is viewed with suspicion, but when it is realised that he is only concerned with a works getting full value from its fuel, rather than to meddle with a particular operating practice, his advice is generally appreciated and applied.

This example of the efforts being made in one district to make the best possible use of coal is typical of work being done throughout the country, but it should not be assumed that nothing had been done in the scientific utilisation of fuel before the war. Actually, the iron and steel industry commenced investigations in 1923 on the subject of high fuel efficiency, and since that time there has been steady progress, with the result that the amount of fuel consumed per ton of product has been gradually reduced.

To achieve high fuel efficiency many works instituted fuel control departments. These were formed to investigate fuel consumption in various stages of production and by control, improvement of plant, and operating efficiency, and, where possible, by installing suitable equipment, were required to improve fuel technque. In many cases remarkable success has been accomplished, indeed some works are operating on a thermal efficiency that is probably not surpassed in any other iron and steel producing country. In these instances no fuel is used other than the coal charged in coke ovens, and, after the production of pig iron, the whole of the subsequent processes in steel-making are met by the use of coke-oven and blast-furnace gases. In general, however, much remains to be done, and the building up of a fuel care conscientiousness will undoubtedly contribute to improved fuel efficiency and result in reducing waste.

Some Recent Advances in the Welding of Aluminium Alloys

By DR. E. G. WEST.

Much work has been done to develop satisfactory methods for welding aluminium alloys, and while the progress achieved is considerable, there is room for further research on the subject to provide data upon which the designer can rely. This article, which is based on a recent paper by the author, given to the Manchester Section of the Institute of Welding, reviews advances made in this field.

TO attempt to deal with every aspect of the subject, or even with the details of any one welding process applied to light alloys, is not intended, nor is it possible within the scope of one article. The present review, therefore, is confined to the wrought aluminium-base materials, and "Recent" has been taken to mean during the last three or four years.

The general problems involved in the welding of aluminium and its alloys have been dealt with previously,[1] and although there are still difficulties with certain types of alloy, it must be noted that all types of light alloys can be welded to-day by one method or another. It is not yet possible, however, to weld all of them satisfactorily by all the processes available, and the present position is summarised in Table I.

TABLE I.
SUMMARY OF WELDING PROCESSES APPLICABLE TO REPRESENTATIVE LIGHT ALLOYS.

Material	Oxy-Gas	Atomic Hydrogen	Metallic Arc	Carbon Arc	"Heliarc"	"Weibel"	Spot and Seam	Butt
Pure Al	R	P	R (T)	N	P	P (T)	P (T)	N
DTD 213A ..	R	P	R (T)	N	P	P (T)	R (T)	P (T)
BSS L46 ...	R	P	R (T)	N	P	P (T)	P (T)	P (T)
DTD 182A ..	P	P	R (T)	P	P	P (T)	P (T)	P (T)
BSS 5L3 ...	N	N	N	N	N	N	R (T)	N
BSS L33 ...	R	P	R	P	P	N	R (T)	P (T)

R = Recommended.
N = Not recommended.
P = Possible, but not extensively used.
(T) indicates that some limit is imposed by the thickness of the metal.

Influence of Properties

It is now generally appreciated that the welding technique which has to be adopted for a specific material depends on the physical, chemical and mechanical properties of the parent metal; consequently a procedure found suitable for one material, say, commercial aluminium, requires modification or possibly complete alteration for a different material, say, the magnesium-bearing aluminium alloys. In order to stress the importance of the proprties of the materials, the comparative figures in Table II are given.

The two most important properties in welding the aluminium-base alloys are first the tendency of the metal to oxidise readily, and secondly the high thermal and electrical conductivities. The effect of other properties is not negligible, and, in particular, metallurgical consideration of the heat-treatable alloys shows that fusion welding cannot give joints equal in strength to the unwelded portions of a structure.

TABLE II.
PHYSICAL PROPERTIES OF ALUMINIUM AND ITS ALLOYS COMPARED WITH SOME OTHER METALS.

Alloy.	Temperature (° C.) of		Mean Specific Heat	Thermal Conductivity (Cgs. Units)	Electrical Conductivity (% Copper)	Coefficient of Linear Expansion (× 10^6)
	Solidus	Liquidus				
ure Aluminium	659	659	0·24	0·534	66	24
SS 2L17	659—650	659	0·24—0·23	0·52	61—65	24
SS 5L3—(Soft)	approx. 550	659	0·21	0·45	50—60	22·9
(Heat-treated)	„ 550	659	0·21	0·28	approx. 30	22·9
TD 213A	655	659	0·21	0·39	42	23
SS L46	600	659	0·21	0·37	40	23
TD 182A	approx. 550	approx. 620	0·22	0·31	35	—
SS L33	577	577	0·20	0·30	37	21·5
ectrolytic Copper	1,084	1,084	0·09	0·98	100	16·4
ld Steel	approx. 1,400	approx. 1,500	0·12	0·11	12	11
ainless Steel	„ 1,400	„ 1,500	0·12	0·03	14	17

This is shown by the figures given in Table III; the properties of cast Duralumin after heat-treatment and ageing compare badly with those of the wrought material.

TABLE III.
PROPERTIES OF A TYPICAL "DURALUMIN"-TYPE ALLOY IN VARIOUS CONDITIONS.

Condition	0·1% Proof Stress, Tons/sq. in.	Ultimate Tensile Strength, Tons/sq. in.	% Elongation on 2 in.
Chill cast	9·2	10·7	1
Chill cast and heat-treated	13·2	18·4	1·8
Wrought and heat-treated	15·0	26·5	18
Wrought and annealed	6·0	17·0	17·8

The effects of these properties on welding procedure must therefore be considered, and it is strongly recommended that explanations as to the reasons for a particular technique should be given to operators. Greater interest is then usually shown in the job, with a consequent improvement in quality.

It is desirable to weld the light alloys quickly in order to reduce the extent of the heat-affected zone, but the thermal properties necessitate a high heat input. In fusion welding, preheating is usual, and resistance welding demands very high currents plus a high degree of precision control throughout the operation.

A major consideration is the influence of the oxide film, which is always on the surface of the light metals and their alloys. On aluminium-base alloys this film consists mainly of aluminium oxide, and on the magnesium alloys it is chiefly magnesia, with some degree of hydration in each case. On the aluminium-base alloys the film is continuous, and it is very tough and adherent. The melting points of both aluminium oxide and magnesium oxide are well above the melting points of the metals, and neither of the oxides dissolves in any of the usual borax-base fluxes. Neither are these oxides reduced to the metal by the welding flame nor by the carbon arc, and it is necessary therefore to use an active flux.

Gas Welding

Aluminium was first gas-welded over 40 years ago, and the mixtures patented[2] then have since been basically the same, consisting essentially of the chlorides and fluorides of alkali metals, usually including lithium chloride. It was generally assumed that some reaction occurred, resulting in the production of cryolite, which dissolved the aluminium oxide.

Work carried out by the author has shown, however, that the mechanism of oxide removal in the case of aluminium is very similar to that in the de-scaling of steel.[3] The solubility of aluminium oxide in molten

1 E.g., *Trans. Inst. Weld.*, Oct., 1939; *J. B'ham Met. Soc.*, 1941, **21** (4), p. 254.
2 "Flux Patents," see *Trans. Inst. Weld.*, 1940, **3** (2), pp. 93–105.
3 E.g., West. "Fluxing Action in the Gas Welding of Aluminium," *Trans. Inst. Weld.*, 1941, **4** (2), p. 50.

flux was extremely small, the highest figure obtained in the author's tests being 3% of pure Al_2O_3 in a good flux at 650° C. Thus it appeared that, contrary to the general belief, gas-welding fluxes for aluminium did not depend for their efficacy on the dissolution of aluminium oxide. A new theory was therefore advanced.

The salts comprising flux mixtures ionise to a considerable extent when molten, and, while aluminium oxide is unaffected by chlorine, the metal itself is readily attacked by these ions giving volatile aluminium chloride which boils at 180° C. As soon as it is produced by this reaction it lifts the oxide layer from the surface. This means that the chloride ions must penetrate the film in order to reach the metal, and they may do this in several ways. Thus, on super-purity aluminium (99·99 + % Al) the flux reaction is very slow, but on commercial purity metal and on many of the alloys the action is rapid, indicating that the more continuous film of oxide on the purer metal prevents access of chloride to the metal surface. The less pure materials have a discontinuous film of oxide, thus allowing the chloride to react with the metal more readily.

Flux mixtures consisting solely of chlorides are not particularly effective, but up to 10% of an alkali fluoride improves the efficiency very considerably. These fluoride ions in the melt react with the oxide, thus allowing the chloride ions to penetrate, but fluorides alone are almost completely ineffective as fluxing agents, because aluminium fluoride is not volatile at welding temperatures.

During fluxing, any alkali metal liberated probably volatilises, but it may affect the properties of the weld (e.g., by "modification" of aluminium-silicon alloys). Gases, whether aluminium halides or alkali metal, passing through the melt probably break up the oxide particles and carry them away from the metal surface to form a suspension. It is not suggested that the same mechanism applies to metallic arc welding in which the higher temperature of the arc may allow solution of the oxide in cryolite to occur.

The mechanism of oxide removal from the magnesium-base alloys appears to be one of solution only, the mixtures most suitable being very similar to those used in the melting and casting of magnesium alloys.

Compositions of flux mixtures are given in Table IV.

TABLE IV.
RANGE OF COMPOSITIONS OF WELDING FLUXES—%.

Salt.	Aluminium and Alloys.		Magnesium-base Alloys.
	DTD 119	Range.	
Li Cl	15 minimum	0—30	8—30
K F	7 „	0—15	0—10
K Cl	44 „	0—60	8—40
Na Cl	Remainder	Remainder	0—30
Others	3 K H SO_4	NaF, $BaCl_2$, etc.	$BaCl_2$, $MgCl_2$, NaF, LiF, etc.

Certain alloys of aluminium are more difficult to flux than others, in particular those containing appreciable quantities of magnesium, such as to DTD 182A–6·5 to 10% Mg. This is apparently due to a high proportion of magnesium oxide in the film. The normal flux mixtures found suitable for aluminium, therefore, require modification before they are suitable for alloys of this type.

On the basis of this theory of oxide removal, it appears desirable to investigate the possibility of gas fluxing. This has been described by Pilling and Kihlgrem[4] for the welding of nickel, for which a suitable fluxing agent was ethyl borate. It seems likely that the use of volatile chlorides, probably organic compounds, might be effective for the welding of aluminium alloys. The welding gas is passed through the volatile liquid, a small amount of which is thus carried to the point of welding. The particular advantage of gas fluxing is the close control which can be exercised over the amount of flux applied.

4 Pilling and Kihlgren. *J. Amer. Weld. Soc.*, 1929, **8** (4), pp. 20–28.

When using the normal powder mixtures the tendency is to add too much flux, which results in undesirable inclusions. Correct methods of application have been detailed,[5] but attention is again drawn to the importance of designing welded components so that flux is not trapped in the joint. Butt welds only are preferred as shown in Fig. 1, but if the design does not permit butt joints only, any lapped joints or fillet welds must be sealed at each end to prevent ingress of moisture, which would react with entrapped flux residues.

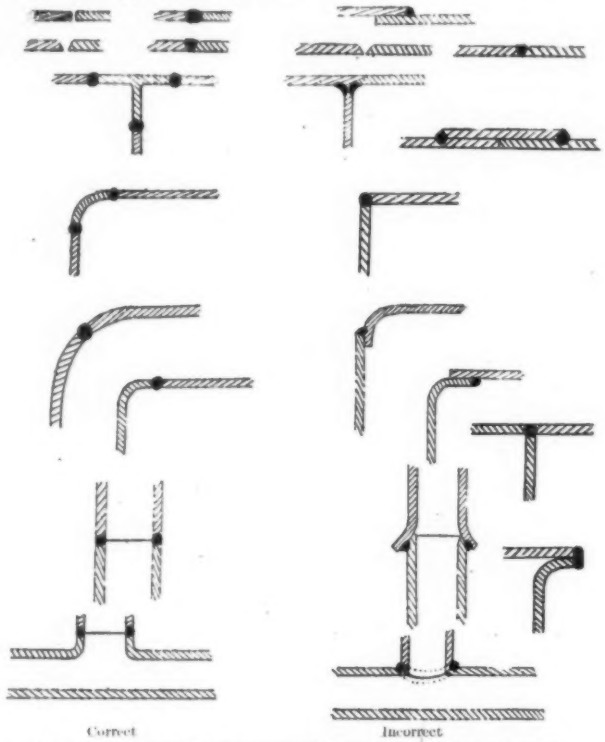


Fig. 1.—Correct and incorrect forms of joint for fusion-welded light metals. (Flux inclusions are entrapped in all joints of the type shown on the right-hand side).

As fluxes are corrosive, all traces must be removed from the weld as part of the welding operation. They are soluble in hot water—hence sufficient washing must be carried out. Following this, treatment in 5% nitric acid may be necessary. To reduce water consumption wash-tanks should be arranged on the counter-flow principle.

Atomic Hydrogen Welding

The use of the atomic hydrogen torch for light alloy welding has not progressed to the extent expected from its advantages. Heat concentration is good, and the flame temperature high, thus allowing rapid welding, but one of the principal advantages of the atomic hydrogen flame when applied to steels—namely, freedom from oxidation—does not apply to aluminium. Atomic hydrogen does not reduce the oxide, and hence flux is still required.

The cost of the process is slightly higher than oxy-acetylene welding, but the use of cracked anhydrous ammonia reduces the cost very considerably. For full advantage to be taken of the speed of this method, automatic operation appears necessary.

Heli-arc Welding

This process, recently developed in America[6] for magnesium alloys, uses a protective envelope of helium with an arc struck between a tungsten electrode (− *ve*) and the work (+ *ve*). The special torch embodies an

5 E.g., Publication TI. "Recommended Procedure for the Gas Welding of Aluminium." Welding Res. Council, 1942.

6 E.g., Loose and Orban. *J. Amer. Weld. Soc.* Res. Suppt., Dec., 1942, **21** (12) pp. 598–607. Anon. *Light Metals*, Nov., 1942, **5** (58), pp. 456–459.

adjustable electrode around which is an annular space for the passage of the helium. Direct current up to 150 amps. is used and welding speeds up to 3 ft. per min. are claimed. Arc lengths in excess of 0·06 in. on magnesium result in poor penetration or gas porosity. The recommended technique is to hold the rod at an angle of 60° to 90° to the electrode and allow the arc to play on to the filler rod, not on to the molten pool. It is stated that for magnesium alloys no flux is required, but it is anticipated that flux will be necessary for the aluminium alloys.

Hydrogen, oxygen, carbon dioxide and hydrocarbons must be absent from the helium, and 7% nitrogen reduces the welding speed to 60% of that obtained when only 2% nitrogen is present.

An advantage claimed for the use of helium is that its specific heat (1·25) allows a valuable quenching effect to be obtained on the surrounding metal. As helium is not available in this country, it has been proposed to use argon, the specific heat of which is, however, 0·12.

Metallic Arc Welding

The metallic arc process is particularly valuable because of the high heat concentration which allows rapid welding, and thus gives a heat-affected zone of negligible width in most instances. The use of a silicon-alloy electrode wire has been general for some years, especially for the repair of castings, but for pure aluminium and the wrought alloys further problems have had to be overcome.

The oxide film gives an unstable arc and necessitates a special flux coating for its removal. Further, arc welds have usually been porous, but recent work has resulted in satisfactory electrodes and technique.

The flux coating differs fundamentally from the compositions used for gas welding, as it must not only remove the oxide, but must also stabilise the arc, but must not melt at too low a temperature. It must also give a readily removed slag.

Some of the salts used in fluxes are hygroscopic andhence coated electrodes must be stored out of contact with the air.

As in the melting and casting of aluminium alloys there are several causes of gas pores in aluminium arc welds—namely, gas in the parent metal or the filler wire, gas or moisture in the electrode coating or adsorbed on the surface of the metal. There are now electrodes fulfilling all the above-mentioned conditions and also capable of depositing sound welds particularly when the parent metal is pre-heated to a minimum of 150° C. before welding commences.

The working data for metallic arc welding are summarised in Table V, some variations being necessary according to the actual job.

TABLE V.

AVERAGE RELATION OF METAL THICKNESS TO ELECTRODE DIAMETER AND CURRENT REQUIRED.*

Thickness of Material (Ins.)	Gauge of Electrode		Current Range (Amps.)	Plate Gap Recommended.
	S.w.g.	In.		
0·062—0·08	12	0·104	40— 55	Close butt
0·081—0·10	11	0·116	55— 65	" "
0·102—0·12	10	0·128	65— 75	" "
0·125—0·15	9	0·144	75— 90	" "
0·156—0·18	9	0·144	90—100	" "
0·187—0·22	8	0·160	100—145	$\frac{1}{32}$ in.
0·25 —0·30	6	0·192	145—190	$\frac{1}{16}$ in.
0·31 —0·37	6	0·192	190—230	$\frac{1}{16}$ in. or bevelled
0·37 —0·45	5 or 4	0·252	230—300	70° with $\frac{1}{16}$ in. step.

* Using as short an arc as practicable.

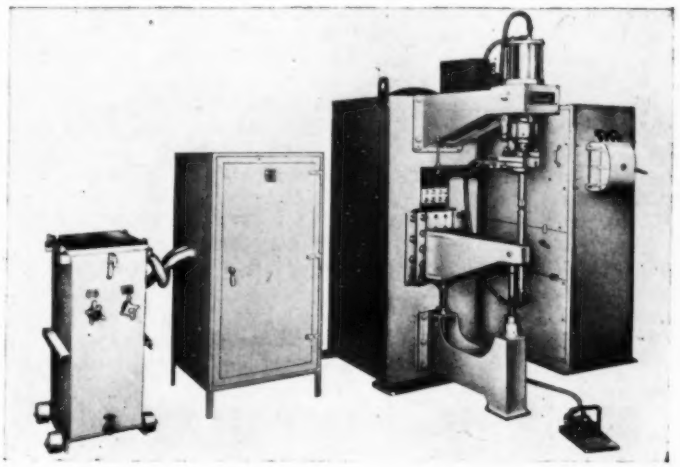
The metallic arc process has been applied to pure aluminium, aluminium-silicon alloys, and the aluminium-magnesium alloys. Its present limit for wrought alloys appears to be the welding of sheet thicker than 12-gauge, but new means of electrical control might extend the process to thinner material.

Carbon Arc Welding

The carbon arc has been used to a small extent, and attempts have been made to employ it as an automatic process. Flux-coated filler rods are usual, and further flux may be required on the underside of the parent metal. It is difficult to avoid porosity and some pre-heating is general.

"Weibel" Process

This process, developed some years ago in Switzerland, possesses promise, but has not attained wide use because of several disadvantages. The heat is obtained by passing a low voltage current via special carbon electrodes through the edges of the metal. Electrodes are of D section, and are held in a spring-loaded torch which allows movement of the electrodes by the operator. The edges are flanged, and the electrodes are placed one on each side of the flanges, which are fluxed on the underside. Current at 4–8 volts is supplied from a tapped, step-down transformer, and as soon as fusion of the edges commences the electrodes are drawn along, to complete the weld. The flanges must be accurate and free from all irregularities, otherwise the electrodes stick and cause excessive melting. It is chiefly for this reason that the process has not been developed, and, further, its application is limited to straight seams, which can be set up accurately on jigs for mass production.



Courtesy Philips Lamps Ltd.

Fig. 2.—Capacity storage spot welder used in the aircraft industry.

The disabilities of this process have been overcome largely in a recently described method, the "Technotherm-Rakos" process, which uses a single carbon electrode applied to flanged edges. The flanging needs to be less accurate than for the "Weibel" method, and the process should be capable of automatic use. The advantage of the Rakos and Weibel processes is in the fast rate of welding obtained.

Spot and Steam Welding

Use of the spot-welding process has developed very considerably in the last few years, dozens of machines now being used where formerly only one was employed. The value of the process from the production point of view is recognised universally, and many improvements have been made by the combined efforts of the machine manufacturers, aircraft constructors, including sub-contractors, and their design and research staffs. The process is being increasingly applied not only to non-stressed components, but also to certain primary-stressed parts.

It is not proposed to discuss the machines now available, nor the welding conditions required for different gauges

of the usual alloys, as this information has been well summarised by several authorities[7] recently. The factors influencing the properties of spot welds may be summarised, however, as follows :—

(1) The current-time relationship at the electrodes.
(2) The pressure applied to the electrodes, particularly its relation to the tip shape and size.
(3) The alloy welded and its condition.
(4) The thickness of the material and the size of the job.
(5) The prewelding condition of the components.

In the author's opinion, the most important aspect to be stressed at present is the preparation of the work before welding.

Preparation involves two fundamental matters, namely—

(1) The fit of the parts before they are presented to the machine.
(2) The surface condition.

Too often, welding machine operators, and sometimes supervisors, attempt to use the welding machine as a press to bring together badly mated parts before the welding operation commences. The force required to place the components in the required close contact decreases the effective pressure exerted by the electrodes and leads to excessive wear of the tips. To-day there is little excuse for placing ill-fitting assemblies between the electrodes as components can be produced to close limits by modern methods of manufacture. The components should be held for welding by tack welds or rivets, clamps, spring-loaded clips, jigs, etc., any holes for fasteners being filled later by rivets.

It is also important to place the work between the electrodes in the horizontal position, to avoid spot welds near the edge and to space them correctly. The oxide film on aluminium-base materials possesses high electrical resistivity, the exact resistance depending on its thickness, mode of formation, moisture content, and the presence of any grease, paint, etc. Such a film is unsuitable for spot-welding, and it is necessary to clean the metal for consistent results, degreasing, usually by a solvent, being the first operation. Mechanical cleaning with emery cloth or scratch brushes is, in general, unsuitable, as the results are inconsistent and there is the danger of removing the aluminium coating from "clad" sheets.

A paste-etch, containing hydrofluoric acid as its active agent, is often applied to the surface and allowed to stay on for a few seconds before washing off with water. It is not particularly satisfactory for unclad alloys containing copper, but it is often convenient for large panels of material similar to DTD 390 or DTD 213A, especially as it allows small areas only to be treated.

For large-scale production, pickling is recommended, one of the most popular solutions containing 5% chromic acid + 15% sulphuric acid (by weight) in water. This is used at 55°–60° C., and is followed by washing, the final swill being in hot water. Pickling time varies from 20 to 40 mins., and this process is now extensively used under close control. An interesting method of control involves measurement of the contact resistance.

The latest solution for pickling employs phosphoric acid at normal temperature, and has been applied on a large scale in America.

Articles should be spot-welded as soon as possible after pickling, otherwise the oxide film re-forms, handling may occur and dirt may be picked up. It has also been shown that moisture adsorbed on the surface is responsible for certain types of porosity in spot welds.

It is necessary to make check tests on spot-welded samples taken at regular intervals. This test should be sufficient to indicate that consistency is being maintained and should be carried out whenever the welding conditions are changed. The usual test is a tension-shear test on specimens containing one or more spot welds.

It is invaluable to carry out metallurgical examination of representative spot welds at regular intervals, and this is essential if cracks or severe expulsion of metal are apparent on the surface.

An important aspect of metallographic testing is to assess the depth of penetration of the weld into the sheets. The optimum figure is usually considered to be about two-thirds of the total thickness for sheets of the same gauge, or two-thirds of the thinner sheet when the two thicknesses are different. The "microtome" method used by the author[8] for weld examination is useful in enabling many specimens, both fusion and spot welds, to be prepared in a short time.

A great deal of fatigue testing has been carried out, most of it in this country being based on a tensile load of P ± ½ P. More research on the fatigue properties of light alloy welds is required and it should be made available in a form suitable for the immediate use of designers. Detailed information on the properties of welds, particularly spot welds, is still one of the most important requirements of the aircraft industry. Work to this end is in hand, largely in this country, by or in close collaboration with the Spot Welding of Light Alloys Sub-Committee of the Welding Research Council.

Additional En Steels

The British Standards Institution has issued an amendment to B.S. 970, 1942, for wrought steels which covers three additional types of steel in the En series. These are :—

(1) *En.* 100, "*Heat-treated Low Alloy Steel Bars for Machining*," the composition of which is selected with a view to advantageous use of the alloy content of steel scrap. It corresponds to D.T.D. 600. The steel is designed to be alternative in all respects to B.S. En. 16, and is regarded as acceptable by all services for the supply of heat-treated bars for machining ordered to Specifications B.S. En. 16, Ministry of Aircraft Production D.T.D. 188A and Ministry of Supply V.9A.

Chemical Composition—

	%
Carbon	0·25–0·40
Silicon	0·50 (max.)
Manganese	1·50 (max.)
Nickel	1·00 (max.)
Chromium	0·50 (max.)
Molybdenum	0·20 (max.)
Sulphur	0·05 (max.)
Phosphorus	0·05 (max.)

The steels governed by the specification are to be supplied in the hardened and tempered condition. Only steels near the lower limit of the composition range may safely be water-hardened unless special precautions are observed. Heat-treatment consists in oil- or water-quenching from 830°–860° C., followed by tempering at a suitable temperature, not below 530° C.

(2) *En*; 104, "*Carbon-Manganese Case-hardening Steel*": covering bars and billets for forgings, forgings and drop-forgings, bars for machining; suitable for a tensile strength of 40 tons per sq. in. min.

Chemical Composition—

	%
Carbon	0·12–0·18
Silicon	0·10–0·35
Manganese	1·25–1·75
Sulphur	0·06 (max.)
Phosphorus	0·06 (max.)

Heat-treatment for test-pieces comprises blank carburising at 900°–930° C.; refining at 870°–900° C., followed by air-, oil- or water-cooling; hardening in oil or water from 770°–790° C. The steel is intended primarily for oil-hardening and it is required that the heat-treatment applied to the acceptance test-pieces should comply with the specification in the oil-hardened condition.

Mechanical Properties (1¼ in. dia. test-pieces)—

U.T.S.	40 tons per sq. in. (min.)
Elongation	20% (min.)
Izod Impact	40 ft.-lb. (min.)

(3) *En.* 102, "*Carbon-Manganese Case-hardening Steel (Free-Machining*," covering bars for machining. Chemical composition is as En. 101, with the exception of sulphur, for which the range is 0.08–0.15%. Heat-treatment requirements are similar to those for En. 101.

Mechanical Properties (1¼ in. dia. test-pieces)—

U.T.S.	40 tons per sq. in. (min.)
Elongation	18% (min.)
Izod Impact	30 ft.-lb. (min.)

8 West. *Metal Industry*, 1942, **61** (21), p. 332.

New Methods for the Examination of Corroded Metal

By F. A. Champion, B.Sc., A.R.C.S.

Two distinct methods of examination are discussed in this paper ; they are included in a new system for the examination of corroded specimens, which has been used successfully for some years, especially for routine work. The effect of the personal element in the classification and assessment of corrosion has been minimised by comparison of the specimens with standard charts and by description of the corrosion in standard terms. The main features of the system have been abstracted from the original report, and are given in this article.*

ACCURATE reproduction of the form and intensity of corrosion, even in the laboratory, is difficult to achieve, and since the accuracy of the method of examination need be little higher than the reproducibility of the corrosion test, the derivation of satisfactory qualitative and quantitative results from the same examination is quite a feasible proposition. With these objects in view a new system of macroscopic and microscopic examination has been devised, and although the completion of the development of a promising method of quantitative measurement was indefinitely postponed on the outbreak of war, in the present state of development, a description of both serve a useful purpose. Indeed, it is hoped that the former system of examination may provide the basis for a universal code for corrosion reports, which would considerably assist the collaboration of different workers and laboratories on current corrosion problems.

The results of the examination by the system described can be given exactly and concisely in correspondence, usually without any necessity for photographs or other illustrations. Primarily designed and used for specimens of aluminium and its alloys up to about 1 dm.² in area and 1 cm. in thickness, the system should be readily applicable to other materials, although some modifications may be necessary.

Macroscopic and Microscopic Examination

The most efficient qualitative examination depends mainly upon the experienced use of the binocular microscope, supplemented, where this first examination makes it appear desirable, by micro-examination of polished sections at higher powers of magnification. Results from these means of examination are affected by the personal factor, and the main purpose of the new system is to minimise the personal element so that reliable, though approximate, comparisons of quantitative, as well as qualitative, results may be made irrespective of the identity of the observer or the time of observations. This has been achieved mainly by the use of standard charts for comparison with the specimen, and by the adoption of standard terms for describing the corrosion observed. These standard terms may be replaced by standard abbreviations whereby the results may be accurately and concisely recorded and tabulated.

General Outline of the System.—Corrosion is normally divided into two main classes, general and local corrosion but for accurate description of corroded specimens further sub-division is necessary. The general scheme of classification follows, with the localisation, and hence usually the danger of attack, increasing in the order given.

- General corrosion
  - Even.
  - Uneven.
- Local corrosion
  - Even.
  - Uneven.
  - Pitting
    - Wide.
    - Medium.
    - Narrow.
  - Cracking.

* *Journ. of Inst. Metals*, Feb., 1943, pp. 47-66.

These different types of corrosion are illustrated in Fig. 1. It should be remembered that the differences between these various types are in degree rather than in kind, and that as corrosion proceeds it may change from one type to another, i.e., the corrosion may change in character as well as in severity with continued exposure.

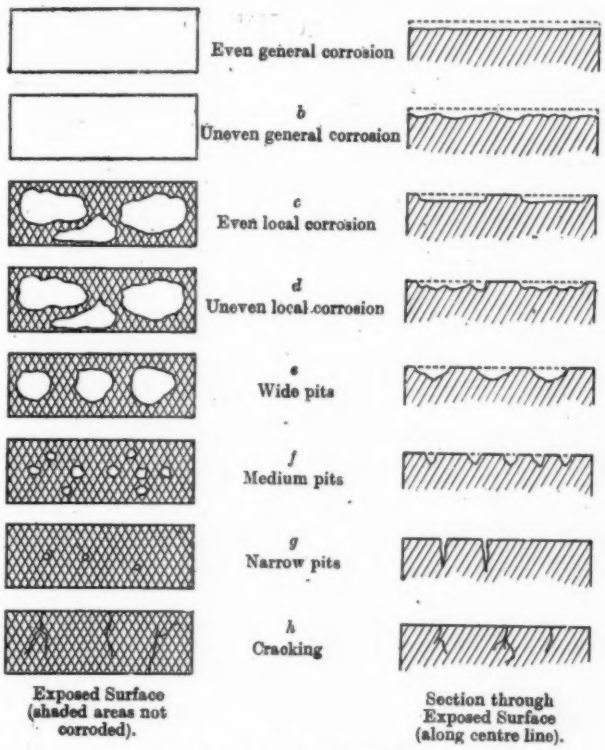


Fig. 1.—Types of corrosion.

When the types of attack exhibited by the specimen have been recognised, the estimation of the degree of attack of each type is then made by comparison with appropriate standard charts. Figs. 2*a* and 2*b* show the charts used for direct comparison with the specimen, and the corresponding standard terms to be sued for expressing the results. Charts A and B are concerned with features in the plane of the corroded surface—namely, the number of sites of attack per unit area (Chart A) and the area affected at each site (Chart B). Charts C and D are concerned with features normal to the plane of the corroded surface, and provide the third dimension in the description of the corrosion by indicating the depth of attack. These two Charts (C and D) cover different ranges of intensity, but are otherwise similar, and in each case the width of the black line gives the depth of attack corresponding to the standard term.

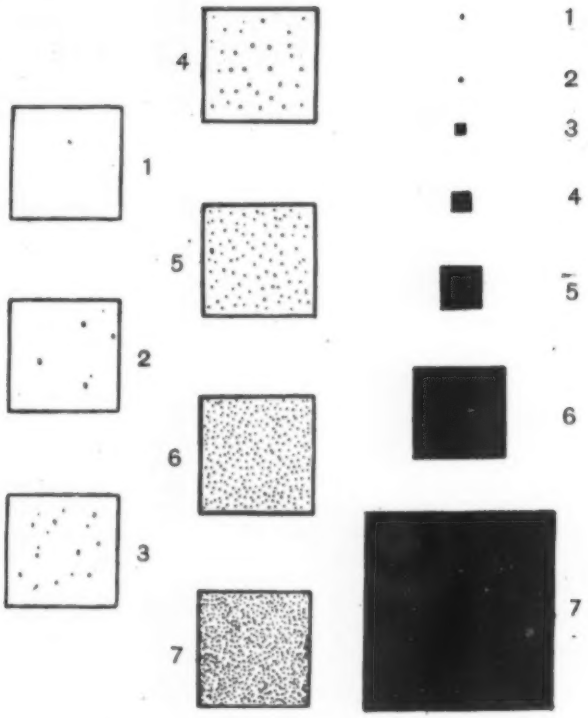


CHART A.—NUMBER.

Key:

1. Very few.
2. Few.
3. Small number.
4. Moderate number.
5. Considerable number.
6. Numerous.
7. Very numerous.

CHART B.—SIZE.

Key:

1. Minute.
2. Very small.
3. Small.
4. Moderate.
5. Considerable.
6. Large.
7. Very large.

Fig. 2a.—Standard chart for macroscopic and microscopic examination.

The charts to be used for comparison with the corroded specimen depend upon the type of corrosion under consideration. General corrosion evidently requires only a descriptive term indicating the average intensity or depth of attack (Chart C). With even and uneven local corrosion, however, it is necessary to define the number of areas of attack (using Chart A), the average size of these areas (using Chart B), and the average depth of attack in those areas (using Chart C). Pitting is adequately described by the number (Chart A) and intensity or depth (Chart D) of the pits if the type of pit (wide, medium or narrow) is given, since those factors, and not the actual area of each pit, are the decisive factors of the metal by pitting. Cracking is described by the number (Chart A) and intensity or depth (Chart D).

The charts are designed to cover the range of attack of aluminium and aluminium alloys experienced in corrosion tests and in articles in service, and consequently it has been found necessary to employ two scales of intensity. General corrosion, and even and uneven local corrosion, have been found to be significant at a depth of penetration considerably less than would be of corresponding importance in the case of the more localised forms of corrosion (pitting and cracking): furthermore, the latter commonly penetrate to a greater maximum depth than the less localised forms of corrosion. For example, Chart D makes no provision for very slight uneven general corrosion such as would affect the appearance of the metal, while it would be misleading to describe a pit equivalent in depth to the width of the fourth line of Chart C as moderate. Separate Charts D and C have therefore been adopted for the intensity of the more and less localised types of corrosion, respectively, as shown in Fig. 2*b*.

The application of the system may be illustrated by the following standard descriptions of the specimens depicted in Fig. 1.

(*a*) Severe even general corrosion.
(*b*) Severe uneven general corrosion.
(*c*) A few considerable areas of severe even local corrosion.
(*d*) A few considerable areas of severe uneven local corrosion.
(*e*) A few considerable wide pits.
(*f*) A small number of considerable medium pits.
(*g*) A few severe narrow pits.
(*h*) A few severe cracks.

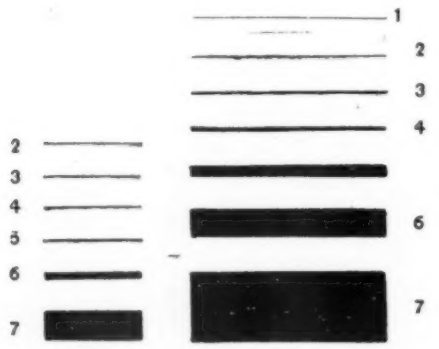


CHART C.

Intensity of general, even local, and uneven local attack.

2. Very slight.
3. Slight.
4. Moderate.
5. Considerable.
6. Severe.
7. Very severe.

CHART D.

Intensity of pitting and cracking.

1. Minute.
2. Very slight.
3. Slight.
4. Moderate.
5. Considerable.
6. Severe.
7. Very severe.

Fig. 2b.—Standard chart for macroscopic and microscopic examination.

N.B.—In these reproductions the widths of the thinner lines are only approximate; the correct widths are given in Table III.

The corrosion of specimens is sometimes more complicated than is suggested by Fig. 1, however; for example, the attack may be partially or wholly intercrystalline in character, or may be affected to some extent by other factors, such as lamination or tearing of the metal. Standard terms have not been adopted to express the influence of such factors, since a certain amount of flexibility of nomenclature is desirable here; but the terms used should wherever possible correspond with those given in Figs. 2*a* and *b* for Charts A, B, C and D.

Standard Abbreviations.—It will be observed from Figs. 2*a* and *b* that the various stages of severity of corrosion represented by Charts A, B, C and D have been numbered 1—7 in each case, and these descriptive numbers may be used to replace the corresponding standard terms. Although no standard terms have been adopted for expressing the influence of other factors on corrosion, descriptive numbers may be used as with Charts A, B, C and D, though with less accuracy. Furthermore, standard abbreviations have been adopted for the standard terms of classification, and for the factors influencing corrosion, and these are given in Table 1. Combinations of the descriptive numbers and the standard abbreviations thus provide concise and accurate descriptions of the corrosion of the specimens.

The complete abbreviated description of each type of attack on the specimen consists of the following, in the order given:—

(i) The type of attack.

(ii) The descriptive number of the attack, as assessed from the four charts of Figs. 2*a* and *b* in the order of number, size, intensity. Where a given chart

is not applicable to the type of attack under consideration the descriptive number may be replaced by a dash sign (—).

(iii) Abbreviations for factors influencing corrosion, where such factors are observed, followed by the descriptive numbers indicating approximately the importance of that influence on the corrosion.

TABLE I.—STANDARD ABBREVIATIONS.

Abbreviation.	Term.	Abbreviation.	Term.
Ge	Even general corrosion.	P	Penetration or perforation.
Gu	Uneven general corrosion.	B	Blistering.
Le	Even local corrosion.	S	Spill.
Lu	Uneven local corrosion.	T	Tearing.
Lw	Wide type of pits.	La	Lamination.
Lm	Medium type of pits.	C	Cavities in metal.
Ln	Narrow type of pits.	E	Extrusion
Lk	Cracking due in part to corrosion.	D	Drawing.
A	Area.	R	Rolling.
Y	Intercrystalline type of attack.	—	No observation made.
Ex	Exfoliation.		

The manner in which these abbreviations are used is shown more clearly in Table II, containing detailed explanations of a few examples and references to the appropriate tables and charts.

TABLE II.—EXAMPLES OF THE USE OF ABBREVIATED DESCRIPTIONS.

Shorthand Description	Abbreviations.	Meaning.	Reference. Table.	Reference. Figure.	Full Meaning.
Ge—6—Y5	Ge	Even general attack	1	1*a*	Severe even general attack, mainly due to intercrystalline attack
	—	(Chart A not applicable)	—	2A	
	—	(Chart B not applicable)	—	2B	
	6	Severe	—	2C	
	—	(Chart D not applicable)	—	2D	
	Y	Intercrystalline	1	—	
	5	Considerable.....	—	—	
Lu243—R3	Lu	Uneven local attack	1	1*d*	A few moderate areas of slight uneven local attack, showing slight association with rolling.
	2	A few areas.....	—	2A	
	4	Areas of moderate size..	—	2B	
	3	Slight attack	—	2C	
	—	(Chart D not applicable)	—	2D	
	R	Associated with rolling..	1	—	
	3	Slight association	—	—	
Ln6—4B2	Ln	Narrow type of pit	1	1*g*	Numerous moderate pits of the narrow type, associated to a very slight extent with blistering.
	6	Numerous pits	—	2A	
	—	(Chart B not applicable)	—	2B	
	—	(Chart C not applicable)	—	2C	
	4	Moderate	—	2D	
	B	Blistering	1	—	
	2	Very slight	—	—	

The use of dashes, to signify charts not used, may usually be dispensed with when familiarity with the system has been acquired, so that the example in Table II may then be written Ge6Y5, Lu243R3, Lu64B2.

Application of the System in Detail.—In Table III are given the details of the standard charts (Figs. 2*a* and *b*) which are used for direct comparison with the specimen, either by the naked eye or under the binocular microscope.

It will be seen that each of the series given in Table III is in geometrical progression, the common ratio of which is given at the bottom of the table. The geometrical was preferred to the arithmetical progression for this purpose, since it permits the relatively small gradations which are required at the lower end of the series. Hence, although the descriptive numbers themselves are in arithmetical progression, the correct average result of the examination of two specimens is not the arithmetical mean of those numbers, but is given by the arithmetical mean of the quantities those descriptive numbers represent, i.e.:

$$a \cdot r^{n-1+x} = \tfrac{1}{2}(a \cdot r^{n-1+y} + a \cdot r^{n-1}),$$

where $a \cdot r^{n-1}$ = the general term of the series,

y = the difference between the descriptive numbers to be averaged,

x = the difference between the smaller of these numbers and the mean result.

Hence:

$$x = \frac{\log\,(1 + r^{y}) - \log\,2.}{\log\,r}$$

The upper limit of y, for which it is justifiable to average results, is not more than 4, and it may be shown graphically that within this limit, and with r constant, the above equation approximates to:

$$x = k \cdot y.$$

When $r = 1{\cdot}5$; the constant $k = 0{\cdot}65$.

and when $r = 5$, $k = 0{\cdot}85$.

The mean value of $k = 0{\cdot}75$, therefore, gives results within the limits of experimental error for all the values of r given in Table III.

Thus, in order to average results, 75% of the difference between the two descriptive numbers (and not 50%, as in ordinary arithmetical means) should be added to the smaller descriptive number, e.g., the mean of Lm42 and Lm62 is Lm5½2.

Consideration of the details of Charts A and B in Table III shows that the sum of the descriptive numbers for "number" and "size," as applied to even or uneven local corrosion, should not exceed 8. If the sum of these numbers does exceed 8, then either there is an error in one or both of the descriptive numbers, or the attack is wrongly classified as local when it should be described as general corrosion.

TABLE III.—DETAILS OF STANDARD CHARTS.

Descriptive No.	Number. Chart A. Standard Term.	Number. Chart A. No. per dm.²	Size. Chart B. Standard Term.	Size. Chart B. Area, cm.²	Intensity. Standard Term.	Intensity. Chart C. Depth, cm.	Intensity. Chart D. Depth, cm.	Factors Affecting Corrosion (% influence)
1	Very few	33	Minute	0·0006	Minute	0·0001	0·004	9
2	Few	100	Very small	0·003	Very slight	0·0004	0·01	13
3	Small number	330	Small	0·016	Slight	0·0016	0·025	20
4	Moderate number	1,000	Moderate	0·08	Moderate	0·006	0·06	30
5	Considerable number	3,300	Considerable	0·4	Considerable	0·024	0·15	45
6	Numerous	10,000	Large	2·0	Severe	0·10	0·4	70
7	Very numerous	33,000	Very large	10·0	Very severe	0·40	1·0	100
Common ratio of series		3		5		4	2·5	1·5

The foregoing descriptions and explanations refer particularly to examinations by the naked eye or by means of the binocular microscope, but the application of the examination of polished sections, etc., by the monocular microscope at higher powers of magnification requires only the preparation of suitable standard charts. Since examination of the charts of Figs. 2*a* and *b* through the microscope is inconvenient in such cases, magnified charts for direct comparison with the magnified image are generally required in order to obtain results strictly comparable with those obtained with the binocular microscope. Where a projection microscope is available, charts engraved on a celluloid sheet have been found convenient. The following points should be borne in mind in preparing such charts:—

Chart A requires to be converted to linear dimensions, and is most conveniently represented on the basis of the traverse of the specimen (independent of magnification), e.g., 2 points per centimetre of traverse represents a considerable number.

Chart B also requires to be converted to linear dimensions, and a separate chart is required for each magnification employed, e.g., at a magnification of × 150 a very small area is represented by 20 cm. of the image. Only the lower range of the series is required.

Charts C and D remain in linear dimension, but special charts are required appropriate to the magnification in use, e.g., at a magnification of × 150 a pit 3·7 cm. deep would be classed as "slight." Again, only the lower range of the series is required.

The table of the percentage influence of factors remains unchanged when applied to the examination of microsections, and is particularly useful in this connection.

Quantitative Examination

Most of the methods used for the quantitative examination of corroded specimens suffer from the inability to differentiate between local and general corrosion, so that they give little or no information of a qualitative character: even correlation with a separate qualitative examination is often difficult and uncertain. The extent of pitting is sometimes measured[1] by removing layers of metal from the specimen by turning, milling, etc., until the pits are no longer visible; the thickness of metal so removed is measured by a micrometer to give the depth of attack. It was felt that if the area of attack could be measured after the removal of each successive layer, and plotted against the corresponding depth below the original surface, the resultant graph would provide a much more informative picture than the mere measurement of maximum depth. Consideration of optical methods of achieving this object led to the use of such methods for the entire process, with the added advantage that no destruction of the specimen is entailed.

The condition of the corroded specimen is first recorded by a radiograph, and, regarding this radiograph as a negative, a series of positives are printed under conditions of extreme contrast and with increasing times of exposure, so that they represent successive sections parallel to the surface of the specimen (analogous to the mechanical removal of successive layers of metal). Photometric measurements of each photographic positive then give both the area of the attack and the depth below the surface corresponding to that positive, although an internal standard calibrated for depth is necessary for the second part of these measurements. No measurements have yet been made on specimens showing actual corrosion, but application of the method to artificial specimens had given promising results, when the investigation was interrupted by the outbreak of war. The advantage of avoiding any destruction of the specimen has been referred to above, and it might even be possible to make approximate measurements on immersed specimens without any interruption of the corrosion process, an advantage of considerable importance in the determination of corrosion-time curves.

TABLE IV.

Diagram.	Total Corrosion.	Mean Depth.	Local Corrosion.	Max. Depth.	$\frac{\text{Local}}{\text{Total}}\%$	$\frac{\text{Max. Depth}}{\text{Mean Depth}} - 1.$	Pitting Index.
1*a*	0·30	0·15	0·00	0·13	0	−0·1	0
1*b*	0·33	0·165	0·00	0·16	0	0·0	0
1*c*	0·205	0·103	0·00	0·11	0	0·1	0
1*d*	0·20	0·10	0·01	0·14	5	0·4	1
1*e*	0·22	0·11	0·063	0·20	29	0·8	5
1*f*	0·105	0·053	0·063	0·19	59	2·6	12
1*g*	0·055	0·028	0·05	0·50	91	17	39
x	0·028	0·014	0·025	0·50	90	35	56

The presentation of informative results in as simple a form as possible is regarded as an important part of the development of any method of corrosion measurement, especially for routine purposes. Accordingly, a working formula has been devised to provide a simple mathematical representation of the corroded area-depth graph referred to above, and this formula has formed the basis of the radiographic method of measurement as developed so far.

Formula for the Expression of Localisation.—A somewhat arbitrary formula has been devised for the calculation of a Pitting Index, whereby the liability of a metal to fail by localised corrosion (e.g., leakage due to perforation) may be expressed numerically.

The liability of a metal to fail in this way is considered to depend on the proportion of the total corrosion which is localised, and on the maximum depth of penetration. The exact definition of local corrosion is open to discussion, but bearing in mind that general corrosion is rarely perfectly uniform, we may regard local corrosion as that occurring below the mean depth of attack, where:

$$\text{mean depth} = k\frac{\text{total corrosion (c.c.)}}{\text{area exposed to corrosion (cm.}^2\text{)}}$$

The value of k should lie above 1 in order to avoid the possibility of including what is really general corrosion (though not perfectly uniform) below the mean depth. The most suitable value probably lies between 1 and 2, but must be determined by actual experiment. For the purpose of this report the value is taken as $k = 1{\cdot}5$.

The localisation of the attack may be expressed independently of the units of measurement as the percentage of the total corrosion which is localised, i.e.,

$$\frac{\text{local corrosion} \times 100}{\text{total corrosion}}$$

The localisation may also be expressed by the ratio:

$$\frac{\text{maximum depth} - \text{mean depth.}}{\text{mean depth}}$$

The values of these ratios corresponding to the diagrammatical sections of Fig. 1 are given in Table IV, from which it will be seen that the first ratio tends to become ineffective when the attack is highly localised, while the second ratio tends to over-emphasise extreme localisation.

The latter difficulties are shown more clearly in the bottom row of figures (*x*), representing a section similar to Fig. 1*g*, but having only one pit. This further increase in localisation has had an appreciable effect on the second but not the first ratio. In order to combine the advantage of both ratios the geometric mean is used for the Pitting Index:

$$\text{Pitting Index} = 10\sqrt{\frac{\text{Local Corrosion}}{\text{Total Corrosion}}\left(\frac{\text{max. depth}}{\text{mean depth}} - 1\right)}$$

The values calculated from this formula are given in the final column of Table IV as an indication of the significance of the Pitting Index. Experimental work is required to determine the most suitable value for the constant use in calculating the mean depth and may suggest other modifications in the formula, but this general type of formula appears to be satisfactory for the purpose in view.

Radiographic Method of Measurement.—A simple radiograph of the specimen is first taken with the corroded surface to be examined normal to the X-ray beam. If the radiograph is regarded as a negative of the specimen, then a series of positives printed from this negative, under conditions of extreme contrast and with increasing exposure times, corresponds to a series of sections of the specimen parallel to and at increasing depth below the surface of the specimen. Using photographic film for the positives, the transparent parts represent corroded areas, while the opaque parts represent intact metal. These positives are then inserted in a suitable photometer, where measurement of the light passing through the film may be used to estimate the corroded area. A standard aluminium wedge placed beside the specimen when taking the radiograph provides an internal standard for the measurement of the depth below the original surface represented by the positive under examination.

The main difficulties in experimental technique were found to be: (*a*) the reduction of non-uniformity in X-ray and light-beams and in photographic recording, and (*b*) the achievement of sufficient contrast in the photographic prints.

A projection printer was constructed for the production of positives in which care was taken to obtain the maximum contrast. The corroded area and depth corresponding to each positive are then assessed from the photometric measurements on the positive. Satisfactory measurements of local corrosion have been made on artificially prepared specimens, but it is considered that the measurement of maximum depth and general corrosion requires further investigation.

The work described was carried out in the Research Laboratories of the British Aluminium Co., Ltd., under the general direction of Dr. A. G. C. Gwyer, B.Sc.

1 G. D. Bengough and F. Wormwell, *Third Rep. Corrosion Com., Iron & Steel Inst.*, 1935, p. 167.

Correspondence

Rubber Die Pressures

The Editor, METALLURGIA.

Dear Sir,

In the February issue of METALLURGIA there appeared a review by Dr. E. G. West on processes used in modern airframe construction, in which the rubber die press and the technique of its application is described.

The author explains in considerable detail the degree of hardness of rubber required and mentions it has been shown that rubber of 50–60 Shore hardness is more suitable than harder rubber. One reason given is that rubber of 50 Shore number hardens to about 80 when under pressure. It is also mentioned that by using the softer rubber increased production has been obtained, and, further, a statement is made that a 500-ton press with soft rubber will do the work which would require a 1,500-ton press if the harder rubber is used. This is obviously a very important point. No data is given, however, to support it, and such definite claims can only be accepted if supported by proof obtained from actual practice. In this respect I venture, in the interests of production, to differ from the author and to correct the fallacy of increased production by using low pressures with soft rubber. Claims made in the article are obviously compiled from theory without foundation in fact.

The use of higher pressures with hard rubber was decided by results obtained from extensive trial and development in the test-room and workshop. Lower pressures were, and are still, used in some cases, but limitations in accuracy, want of conformity to pattern shape, and low production figures determined that the low pressure had a limited scope and should be discontinued.

First let us consider the flanging operation: the bending moment exerted by the rubber necessary in practice to form a flange of given width and thickness of metal can be found by simple calculation, narrow "lip" flanges require more pressure than wide flanges of the same metal thickness. Since the pressure per square inch decided the value of the bending moment, it goes without saying that the so-called high pressure gives a much greater range of effective flanging. Further, if soft rubber is used to form a narrow flange a condition can arise where the deformation of the rubber, when under pressure, will cause it to flow in advance of the bending of the flange and get trapped between the former and the metal, thus preventing the flange from being formed.

Joggles often occur in airframe flanges; these require considerable pressure; with pressures of less than 1 ton per sq. in. they cannot be defined, and if the pressure is not high enough, they must be made by hand; this entails a second operation.

Reference is also made to "backing-off" angles to allow for spring back. This, even if the results were uniform, is surely a makeshift method to obtain what is accomplished by higher pressures in one operation. The pressure required to give definite angles and details must have a reserve sufficient to make allowance for the different degrees of heat-treatment and corresponding variations in physical properties of various batches of material.

There is no controversy, it is not a matter of opinion that has decided the higher pressures now accepted. Decisions have been based on results obtained from actual production practice in the factory.

Referring to increased production from low-pressure rubber this is entirely wrong, in fact when soft rubber dies are used part-forming has to be done first; this entails two operations and two sets of dies. Cases have occurred, involving large numbers of dies to part form and finish form on a low-pressure press, where, when a high-pressure press was made available with a hard rubber die, the parts were formed in one operation instead of two, thus cutting down production time by 50%.

Reference is also made to the method of holding the rubber in its pressure box, and it is suggested that anchorage by immersed nuts give the rubber freedom to flow. Surely it is obvious that the effective depth of rubber available for displacing is reduced by this method of fastening. There is no point in having about 2 in. of rubber or the depth of immersion of the nuts or plates idle. A simple countersunk perforated plate only absorbs about $\frac{1}{2}$ in. of rubber and leaves the entire block available for displacement. The question of flow does not arise, since displacement takes place almost entirely where forming takes place. The rubber presumably fills the pressure box, and the amount of displacement at the point of anchorage is infinitesimal. The rubber should, in fact, be the size of the box, and flow is only possible with a more or less incompressible fluid, when free space is available and side displacement. Possibly this represents idle motion, no useful "coining" effect is accomplished until the rubber forms a solid mass completely immersing the parts being processed.

The application of the "rubber die" press is comparatively new, and its use will undoubtedly be extended as it becomes more widely known. It is important in the early stages of description to proceed cautiously and establish design and technique determined by actual use.

G. B. BURNSIDE.

London, S.W. 1,
March 19, 1943.

To the Editor, METALLURGIA.

Dear Sir,

I have read with regret Dr. West's reply to my letter, published in your March issue, in connection with that portion of his article in your February issue which deals with "Pressing with Rubber," and note that he has not dealt with several points raised in my criticism. After carefully reading his reply and the reference to the subject in his original article, I am convinced he is theorising.

Apparently, he agrees with one point, that greater pressure is undoubtedly necessary for thick gauges, and then goes on to say that a great proportion of current production can be more efficiently obtained using soft rubber. I cannot accept such a statement without some supporting evidence, which should make interesting reading when it becomes available.

Regarding the life of rubber pads, information in terms of life, rather than on the number of like components produced, is available. I have first-hand knowledge of a 9 ft. by 6 ft. hard rubber pad, working at high-pressure intensities, which has operated satisfactorily over a period of three and a half years; I doubt whether a soft rubber pad, held in the way Dr. West mentions would not sag and cease to be useful in much less time.

In his original article he stated that, for large components combined blanking and forming could be carried out economically. I would like to know what grounds this statement is based upon. Blanking can be done, but, in my experience, the cost in time and money is greater than routing or blanking by means of "push through" type of tools, and it is extremely difficult to obtain clean-cut edges on the rubber press.

Finally, he refers to outputs of "over 40,000 parts per day";* has he ever seen a press operating at this rate for more than one or two closures, or has he analysed this statement, and discovered that in a day of two "10-hour" shifts, and assuming 10 tools in the set up (a fair average), 200 closures per hour would be required, and that the most modern high-speed hydraulic presses only have a cycle of 55 secs. ?

* We have examined the original script and find that a typographical error occurred in the article as published; the number of parts should have been 10,000.—EDITOR.

It appears to me, Mr. Editor, that, in the past, much has been written on this excellent process which has been misleading, and has led to its inefficient operation.—Yours faithfully,

R. F. TAYLOR.

Research Department,
A. V. Roe and Co., Ltd.,
April 2, 1943.

The Editor, METALLURGIA

Dear Sir,

I thank you for allowing me to see the further correspondence on my statements regarding the hardness of rubber bolsters used in hydraulic pressing. It should be unnecessary to point out that my original contribution was a brief review covering a number of aspects of the subject, and it was obviously impossible for me to do more than touch on a few topics. I regret that at the moment pressure of work prevents me from dealing with this interesting and most vital matter in the detail it deserves, although I hope to do so in the near future.

I cannot accept the statement by your correspondents that the original article was "compiled from theory without foundation in fact." Although I have not had a rubber press under my charge, as Mr. Taylor has, I have had placed at my disposal a great deal of accumulated information and experience by some of the leading rubber press users in the country, and my conclusions regarding the hardness of the rubber for maximum production are based on these data. This, I submit, gives more weight to my statements than is credited by your correspondents, one of whom admits having operated a press with a hard rubber pad for three and a half years, but does not mention whether he has any first-hand information regarding the use of a soft rubber pad. You will appreciate, however, that as I have not been personally responsible for the experimental work on which I based my original statements, I cannot, unfortunately, give details, although I believe this will be possible in due course.

I believe that the whole point of this controversy is that rubber must be regarded virtually as a fluid, and this is the basis of statements made by myself and others that rubber of 50 Shore hardness when under pressure behaves like rubber of 80 hardness. Thus, if the rubber is hard before pressure is applied, a greater total pressure is necessary to make the rubber flow freely. Leading rubber technologists (not necessarily press experts) agree that compressed rubber behaves fundamentally like a fluid, and I contend that the inefficient operation of rubber presses has been due to the practical man attempting to use rubber without realising this fundamental point—sometimes with disastrous results to the walls of the rubber box or to the head of the press.

Within the last year there have appeared in other journals several reports giving valuable data on the use of soft rubber, including criticisms of articles by Mr. Burnside.

That recommendations to use hard rubber are not always accepted is shown by an article in *Machinery*, dated August 6, 1942, which states that a 5,000-ton press has just been installed with a rubber pad of 50 to 55 Shore hardness, with a considerable saving in power.

I regret that in the original article a printer's error was responsible for the statement that rubber presses were producing 40,000 parts per day. If I had given the figures in detail it would have read 10,000 parts per day of two shifts.

Finally, Mr. Editor, I submit that the only way to continue increasing production with decreasing man-power is to investigate all new proposals, trying out new ideas and techniques with unbiased minds and discussing the results frankly in an impartial manner. If my original article and, the subsequent correspondence does this for rubber pressing, it will have more than achieved its object.—Yours faithfully

E. G. WEST.

Birmingham,
April 7, 1943.

Measuring and Testing Thin Tin Coatings

THE present need for tin economy has brought into prominence a new product, electro-tinplate, consisting of steel sheet or strip carrying 8 oz. or less per basis box of electro-deposited tin. This material differs from the traditional hot-tinned tinplate in that the tin coating is generally much thinner—about one-third that of normal tinplate—the structure of the coating is different, and there is no considerable tin-alloy layer between the tin and steel. In many cases, however, the electro-tinplate is subjected to a thermal treatment which momentarily fuses the tin coating; an alloy layer is then formed and the coating has some of the properties of a hot-tinned coating.

This change in normal practice in view of existing conditions has caused a number of technical difficulties; it is important, for instance, that manufacturers should be able to assess the thickness of the tin coating that they are applying, and this has hitherto presented some difficulties. The full range of methods available is given in Publication No. 115 by the Tin Research Institute. The methods described come under the following general headings:—

(*a*) Weighing the component (or test-piece) before and after stripping the tin in a chemical solution.
(*b*) Completely dissolving the specimen and determining the tin by analysis.
(*c*) Measuring the time required to dissolve the coating.

The coating on the new electro-tinplate is so thin that the established methods of testing its continuity are inapplicable. This is an important factor, concerning the corrosion resistance of thin coatings and its assessment necessitates new technique. Dr. R. Kerr has described experimental technique, which is found to be applicable to these thin coatings, and this description is given in Publication No. 116 by the Tin Research Institute. One of the new tests described is particularly interesting, as it provides a simple numerical value of the amount of iron exposed to attack over a considerable area, without making a laborious count of rust spots and without the elaborate apparatus required for the hydrogen test.

The hot-water porosity test and the ferrocyanide paper test, designed for hot-dipped tinplate, have been modified so as to be applicable to electro-tinplate carrying a much thinner tin coating. In the first of the methods, which is suitable either for thick or thin coatings, an oxide film is formed on the surface of the tin before applying the hot-water test. This pre-filming treatment greatly accelerates the test, so that clearly defined rust spots are developed in 40 mins.

The second method is a modification of the ferro-cyanide paper test. A paper of suitable texture is soaked in a ferrocyanide solution containing a textile penetrating agent, and is then pressed on to the specimen with a roller squeegee. After 10 mins. the pores on the coating appear as blue spots on the paper.

The third test determines the quantity of iron dissolved from pores in the tin coating when the specimen is immersed for 15 mins. in an acid thiocyanate solution. This method gives a measure of the quality of the tin coating, and is applicable to any type of tinplate.

Copies of these publications—Nos. 115 and 116—may be obtained free of charge from the Tin Research Institute, Fraser Road, Greenford, Middlesex.

Mr. Wragge Appointed to the Directorate of James Mills

Mr. W. B. Wragge, B.Sc., has been appointed a Director of the Exors. of James Mills, Ltd., Bredbury Steel Works. In view of the tremendous amount of work Mr. Wragge has done in recent years, particularly in connection with lead-bearing steels to facilitate machining, this appointment is richly deserved.

Rationalisation in Iron Foundries

Part II. *(Contd.)*—Application of Principles

By F. L. Meyenberg, M.I.Mech.E.

In the last issue the author considered the analysis of time, and referred to various additions for lost time which must be considered in making such an analysis. In the present and concluding section attention is directed to making work and time studies and fair payment, and a final section briefly reviews costing studies.

Making Time and Work Studies

IT will be obvious, from what has been explained about work analysis and time analysis, that these studies are single observations to be evaluated and applied afterwards individually, but that each study consists of various stages, and the results of various studies must be connected in order to obtain the full benefit from this research work. Thus, the following stages of the studies can be distinguished :—*

(1) *Analysis of Work.*—The operations are examined and analysed as far as is thought desirable. The various times required for the operations or motions are measured, special attention being given to delays and lost time.

(2) *Rationalisation.*—Any improvement in the methods of working suggested by the information thus obtained is carried out. This includes the elimination of delays and lost time, or wasted and unnecessary motions, speeding up of operations, and any alteration of the machine or tools required.

(3) *Time Study.*—Measuring the times necessary for the improved or rationalised process, and recording them in a prepared form.

(4) *Analysis and Criticism of Time Study.*—Further examination of the results of the time study with a view to investigation of the possibilities of further improvement. Criticism and, if necessary, separation of unsuitable or doubtful figures. Calculation of piece-work rates, if necessary, or performance standards.

(5) *Further Utilisation of Results.*—Preparing the system of tables for the purposes previously mentioned and entering in them the relevant figures—process or operation times—from the time-study forms. Testing and proving these figures by further studies.

The extent to which it is worth while to carry out any or all of these stages depends both on the nature of the work and its magnitude ; that is to say, the number of times the processes and operations are repeated and the possibilities of improvement revealed. Only the man who makes the study can decide this, and he must keep a reasonable proportion between the possibility of achieving results and the amount of study involved in the research.

It has already been mentioned in the general part of this report† that four main aims of work and time studies can be distinguished : (1) improvement of production, (2) determination of a fair payment to the workmen, (3) connection between production on the one hand and costing and accountancy on the other, especially the development of standard performances as basis of standard costs, (4) collection of psychological data. It has further been mentioned that of these four items, the second, has been, and is still in some quarters, considered the most important if not the only valuable result obtainable from time studies. Although the writer of this report dissociates himself expressly from this point of view, it may provide a justification for dealing in greater detail with this second item, the determination of a fair payment of the workmen, especially as in practice there are many wrong ideas on how this aim can be attained.

Fair Payment of Workmen

This discussion deals only with the piece rate system, since the fixed hourly rate is independent of the work done in the time paid. All other wage systems known as incentive, premium bonus, etc., systems contain, in principle, an unfairness to the workmen if they are not used merely as additional to one of the first-mentioned systems in reward for saving material, avoiding rejects, etc.—i.e., generally speaking, for diminishing waste.

It may first be repeated that time studies are only one, though perhaps the most important of the methods used in foundries to determine fair piece-work rates. The others are estimation, use of data of experience and comparison. It must be stated however that these do not constitute four strictly separated methods, only one of which can be applied in a special case ; quite the contrary is correct, and the limits between them are so fluctuating that the transition from one to the other is sometimes almost imperceptible. Moreover, the last method, comparison, is based on figures obtained by the three others. Thus the above distinction is made rather for making the principles more readily understood than for practical purposes.

What is meant by the word "estimation" in this connection can be taken from the following notes :—

(1) Estimation is the most inaccurate method of determining the working time of a job ; it should be applied only where more accurate methods cannot be used either for lack of time or for economical reasons.

(2) Estimation should not be applied in series or mass production, as errors of estimation would be multiplied correspondingly in these cases.

(3) Estimating in the bulk is inadmissible ; the result would not be accurate enough.

(4) The analysis of the work—as explained before in detail—and the estimating of the times for the individual parts of the work increases the accuracy, as long as the subdivision of the work is not carried too far. In this case an accumulation of many small mistakes can result in useless figures for the total work.

(5) It is sometimes easier to estimate the output per shift than the working time of an individual job.

(6) Estimating must be carried out as carefully as possible ; the fact that it is less accurate than other methods should not lead to carelessness.

In order to explain item (5), and to give at the same time time an example of the method of estimating, the following figures of a case taken from actual practice may be quoted :—

Shift-time = 8 hours = 480 mins.

Adjustment—supplementary—and lost times per shift, adjusting the moulding machine, and warming up the pattern plate	10	mins.
Fetching sand	12	,,
Fetching cores	6	,,
Waiting for iron and casting	75	,,
Putting work-place in order	15	,,
Receiving orders, discussion with foreman, relieving nature, etc.	32	,,
	150	,,

480 − 150 = 330 mins.

Output in 330 mins. estimated to 60 moulding boxes,

piece-time per flask $\frac{480}{60}$ = 8 mins.

* The following six paragraphs are taken from "The New Management," by Hildage, Marple and Meyenberg. Macdonald and Evans, London, 1938.

† METALLURGIA, January, 1943.

When the working times are no longer taken directly from memory, or from a rough estimate, but from tables of systematically collected and criticised figures, the estimation turns to a use of data of experience. Some people consider the time used in the past for the same, or similar, work directly as such data of experience and take them from the wages tickets without criticism. Readers cannot be too strongly warned against this method. Only when it is possible to evaluate the times on the wages tickets based on an accurate knowledge of the actual working conditions during the work, and to remove all delays caused by special circumstances, etc.—and that is possible only in rare cases—can these wage tickets be helpful for the collection of data of experience. It is, of course, another matter if somebody familiar with all special circumstances—as, for instance, the foreman—considers these wage tickets during and immediately after the finishing of the job in question and derives from them the data required. If they are then collected systematically, and the unavoidable fluctuations have perhaps been eliminated by the comparison method, these data can become a very useful basis for determination of working times; in some cases they have even been developed to a part of an agreement between the organisations of employers and workmen, as in the following example :—

FIXING OF CORES IN FOUNDRIES FOR MASS PRODUCTION.

(The figures, actual working minutes, are valid for small, horizontal cores, which can be fixed without special difficulty.)

Number of Cores per Mould.	Number of various kinds of Cores.				
	1 to 2.	3 to 4.	5 to 6.	7 to 8.	9 to 10.
1 to 5	0·18	0·20	0·22	—	—
6 to 10	0·35	0·40	0·42	0·47	0·50
11 to 15	0·54	0·60	0·64	0·70	0·76
16 to 20	0·72	0·79	0·86	0·93	1·01
21 to 25	0·90	0·98	1·08	1·17	1·26
26 to 30	1·08	1·18	1·29	1·40	1·51
31 to 35	1·26	1·38	1·51	1·64	1·75
36 to 40	1·44	1·59	1·72	1·87	2·01

The comparison method is, as already mentioned, based on figures obtained by one of the three other methods. Such figures for similar jobs of various sizes—e.g., belt pulleys of equal design, but different dimensions, are collected in tables, or preferably graphs, and intermediate values determined by interpolation. It is possible to compare total working times, as well as partial working times, according to the analysis of time as explained beforehand. Generally, the comparison of partial times can be recommended as more accurate. The most important problem is the choice of the item to which the time should be referred, or—if only graphs are taken into consideration, and as usually the time is taken as ordinate—the choice of the abscissa.

This question can be decided only in the individual case, as the decision may depend on shape and size of the patterns, size of the moulding boxes, etc. Many attempts have been made to find out a relation between the total time and a peculiarity of the object to be moulded, and it seems to be very probable that the connection between surface of the pattern and total time can, if all other working conditions remain equal, be represented by a curve, the course of which could be determined, if three or four points have been found by actual practice. But these results are not certain enough that general conclusions could be obtained, and at least at present nothing else seems to be possible but the investigation of each individual job using, however, such partial times as have been found already for other jobs, and are equal in the new one. If the analysis of time has been carried through to operations or even to motions, and the partial times for these operations or motions have been determined in a series of cases, it is, fortunately, possible to use them also for the new job, as operations and motions of the same kind are repeated again and again in foundry work. Thus, the work of determination of fair working times actually means the development—by means of estimation, use of data of experience, time studies and comparison—of tables of these partial times which can be used as standard times and be combined to give total working times.

Examples taken from actual practice in foundries will show more clearly what is meant by these general explanations.

Machine Moulding Shop Example

As large numbers of a peculiar design of casting are usually produced in machine moulding shops it pays to base the standard times mainly on time studies, especially as the number of standards is not so large as might be expected from superficial consideration. For the usual moulding machines perhaps twenty-one various processes, operations and motions must be covered by such standard times from which the total working time can be determined : Setting up the moulding box (1) ; clamping the moulding box (not always necessary) (1) ; dusting with parting sand (1), setting up the gaggers (when necessary) (1) ; filling in the facing sand (1) ; pressing on the facing sand (1) ; shovelling in the backing sand (1) ; striking off the backing sand (2) ; preliminary ramming by a peen rammer (3) ; flat ramming (4) ; venting by piercing the rammed sand (1) ; turn over the drag or cope and plate (1) ; rapping, vibrating or jarring (1) ; lifting off or stripping (1) ; putting aside (1) ; fixing cores (1) ; adjusting the gate stick (1) ; making up the gate (1) ; building up the pouring basin (1) ; loading or other means for keeping box parts together (1) ; casting (1).

The functions marked (1) may occur with moulding machines of any kind, function marked (2) only with moulding press, marked (3) with hand moulding machines, as lifting, stripping and turnover machines, in special cases also with moulding presses, and marked (4) with hand moulding and with jarring machines.

Special graphs or tables have been developed for each of these functions, which may consist of a combination of various motions, etc. For example, building up the pouring basin may consist of—putting a gate stick into the gate, setting a bush or ring on the cope round it, filling in and pressing sand at the same time forming the basin in the bush, removing the gate stick and smoothing over the edges of the sand. The time (in minutes) necessary for this function can be taken from the following table :—

Diameter of the Bush in Inches.	Height of the Bush in Inches.			
	2.	3.	4.	5.
6¼	0·58	0·70	—	—
8	0·55	0·58	—	—
9½	0·60	0·70	0·85	—
11	0·75	0·80	0·90	1·05
12½	0·75	0·85	0·95	1·10

Into what detail these basic investigations must be carried may be illustrated by the consideration of the determining factors for riddling facing sand. The time depends not only on the quantity of the sand and the mesh of the riddle, but also on the quantity of sand shovelled at once into the riddle. If several shovels of sand are filled in at the same time, the sand balls and falls considerably slower through the riddle than if only one shovel has been filled in and the next not before the sand of the first has nearly fallen through. This is one of small and obvious facts, the neglect of which increases the working time without facilitating the work of the man.

The total working time of a job in the machine moulding shop could be built up by addition of the individual times of the before-mentioned functions in the correct manner, but this calculation can be simplified considerably for practical purposes—for instance, the total working time for comparatively simple moulding boxes has been determined and taken as standard ; all surplus work originating from larger size or difficulty of work is added according to special calculations, e.g. :—

Drag: Time for a standard moulding box, 16 in. × 16 in. × 16 in. deep	= 2·11	mins.
Surplus work—riddling on shovel of facing sand, difference 0·42 — 0·22	= 0·20	,,
Preliminary packing of facing sand	= 0·32	,,
Twice ramming the sand in the box part—30 in.	= 0·07	,,
Venting by piercing the rammed sand 4 in. × 10 in., dist. ¾ in.	= 0·37	,,
Total drag time	= 3·07	,,
Cope: Time for a standard moulding box, 16 in. × 16 in. × 5 in. deep	= 1·69	mins.
Setting the gate stick	= 0·06	,,
Twice ramming the sand in the box part	= 0·07	,,
Cleaning out the gate	= 0·19	,,
Total cope time	= 2·01	,,

Drag and cope = 5·08 mins.

and additions for lost time, falling off in the rate of working, etc. If this method is applied in connection with time studies and checked continually by samples taken off-hand, very reliable working times can be obtained which can be used as a basis for piece-work prices and will be considered as fair by the men, as experience has shown in many cases.

A further simplification of the determination of working time in a machine moulding shop can be obtained by classifying the whole work into, say, three groups :—

(1) *Simple.*—Few and simple corners, compact portions, joint in one plane.

(2) *Medium.*—Several corners, ribs, etc., slim portions, not deep, but necessitating strengthening by nails ; joint not in one plane.

(3) *Difficult.*—Many corners, ribs, bosses, cavities, etc., slim portions, deep and narrow slots, necessitating supports for sand, joint very irregular, complicated shape rendering lifting difficult.

Now a standard flask for each of the three groups must be developed, and if the fixing of cores is taken into account separately the determination of the working times of the majority of the jobs has been reduced to the addition of a few figures taken from graphs or tables. Thus, for example, it has been found in a machine moulding shop that for an ordinary lift machine using moulding boxes 16 in. × 16 in., the following times in minutes apply :—

	Depth of Box Parts. 10 in.	14 in.
Simple work, less core-fixing	5·0	5·5
Medium work, less core-fixing	5·25	5·75
Difficult work, less core-fixing	5·65	6·15
Additional time for fixing simple cores:—		
One core	0·10	
Two cores	0·17	
Three cores	0·25	

Especially difficult moulding work must, of course, be paid separately by a corresponding addition of the resulting time, if a higher flat hourly rate is not preferred in such cases. The same applies to the fixing of complicated cores.

As this report is only intended to give the idea how the determination of working times can be and has been carried out, these examples of jobs in the machine moulding shop may be sufficient, and it need only be added that hand-moulding, core-making, cleaning and dressing can be treated in a similar manner, of course adapted to the different working conditions in these shops.

One further example, however, of the exclusive use of data of experience may be mentioned in order to emphasise that the planning and rate-fixing engineer should not make a fetish of the time study method, but should use whatever method seems to be best for the special job at hand. The figures of the following table refer to the service at a cupola with a sloping hoist ; they are based on careful statistics, and proved correct in practice. Of course they can be used directly only for the plant where they are developed, but may be interesting because of their construction and for comparison purposes.

It should be obvious from these explanations that the introduction or reorganisation of the determination of fair piece times or working times, according to the ideas discussed, is not an easy and short-term job. Further, it

CUPOLA WORKING TIME.

Work.	Basis.	Working Time in Minutes at a Charge of: 10 tons.	15 tons.	20 tons.	25 tons.	30 tons.
Smashing and loading the pig iron, leaving the wagons at the weighing machine.	20% pig iron, 45 mins. per ton.	90	135	180	225	270
Smashing scrap, but leaving it on the ground.	80% scrap, 10 mins. per ton.	80	120	160	200	240
Carrying off remainder of the coke and the slag; repairing the melting zone.	—	360	380	400	420	440
Charging bed coke.	6 wagons, 3 mins. each.	18	18	18	18	18
Charging pig iron and coke.	Up to 20 tons, two men, beyond 20 tons three men, period 5 mins.	200	300	400	500 (250) (coke trans-porter)	600 (300) (coke trans-porter)
Lining the ladles.	240 mins.	240	300	360	420	480
Tapping, preparing settler and slag pocket.	One man.	525	525	525	525	525
Dropping the furnace.	Three men, 30 mins. each.	90	90	90	90	90
Waiting times.	—	90	90	90	135	135
Total working time in mins.		1,693	1,958	2,223	2,783	3,098
Minutes per ton		169·3	131	111	111·5	103
No. of men, taking the day to 525 mins.		3·22	3·75	4·25	5·30	5·90

should not be overlooked that even an expert of long experience needs some time to make himself familiar with the peculiar conditions of the work in question before he can start the actual work, otherwise dangerous mistakes, especially from the psychological point of view, may result which can render doubtful the success of the whole work.

Costing Studies

The fundamental importance of costing studies in general, as well as from the point of view of rationalisation in particular, has been pointed out already in the first part of this report. It may, however, be of interest to present some examples based on principles which are accepted as correct and provide the basis of calculations in iron foundries. As the figures quoted are presented only as examples, they may be changed, according to actual conditions existing, for use in a special case.

It is well known that the price that can be obtained in the market does not depend only on the costs incurred in producing, selling and distributing the goods in question ; the laws of supply and demand play an important role. In the long run, however, the costs will mainly determine the price, or at least its lower limit, as no producer can sell his products for any length of time at a price lower than his costs. On the other hand, it is likewise well known that if somebody uses a wrong calculation of costs and therefore puts his selling prices too low, it may take months before this becomes obvious ; during this time the market can be influenced so badly that not only he alone but the whole branch of industry will come to harm. It is, therefore, of the highest interest that everybody uses "correct costs." Hence the question arises what are the correct costs as a basis of the selling price ? The source mentioned in the beginning of this report, answers this question by recommending costs calculated as follows :—

Costs of molten metal. + Productive or direct wages. + Producton* overhead charges. = Total of production costs.
*200% of productive wages.

Production costs. + Selling and* distribution costs. = Total cost.
*18% of production costs.

Total cost. + Addition for* risk and profit = Minimum selling price.
*12% of total costs.

Minimum selling price. + Special additions for high-grade pig iron, alloys, or high risk of rejects. = Selling price in special cases.

It is, of course, questionable whether this is really the best scheme of cost calculation. As it is almost impossible to obtain a complete agreement on this very much disputed problem, this scheme has been selected as standard throughout this branch of industry in one highly industrialised

country, and has proved useful in so far as a uniformity and comparability has been obtained. It is, of course, necessary to determine continuously from month to month how far the overheads and additions obtained actually in the works differ from this scheme in order to know the true economic position.

Some explanatory remarks concerning the individual positions of the above scheme may be useful, thus :—

(1) *Costs of Molten Iron* are all costs that have arisen up to the moment when the fluid metal coming from the cupola furnace is in the ladle, i.e., costs of pig iron and cast scrap, as well as additions for special mixtures, such as ferro-silicon, etc., bed-coke, melt-coke, limestones and other melting materials ; scale, waste, and general risk of rejects, wages for breaking of pig iron, charging, melting and tapping ; freights, transporting and unloading wages for the materials before-mentioned ; maintenance of the cupola furnace—materials and repairs ; consumption of power of the blowers, charging hoist and pig breaker ; costs of analysis in the laboratory.

Charging of pig iron, scrap, including freights, etc., on the one hand, and melting costs proper on the other, are calculated separately, as the melting costs can be taken as an average for all mixtures, while it is recommended to fix the charging costs proper separately for each mixture to be produced in the works in question.

The average risk of rejected material has been taken as 8 per cent., the scale in the melting process to 7%. Discards and waste of any kind, i.e., gates, runners, risers, lost heads, losses in transport and in the stockyard, etc., should be taken into account for the individual jobs. For cases where that is not possible, average figures are given for castings of various weight groups between 2 lb. and 11,000 lb. per piece.

When using these figures and, as example, a standard mixture for usual machine castings, the following table can be compiled :—

Group No.	Weight of an Individual Casting in Lbs.	Charge to Furnace per 100 lb. Finished Goods.	Costs of 100 lb. Molten Iron in % of the Costs of Group 1.
1	2¼	330	100
2	4½	249	86·3
3	11	212	80·0
4	22	185	75·5
5	55	164	71·8
6	110	148	69·0
7	220	139	67·7
8	550	134	66·5
9	1,100	129	65·8
10	2,200	126	65·3
11	4,400	124	65·0
12	11,000	123	64·8

This table should be the best proof of the necessity to take into account, as carefully as possible, the quantity of waste, and to use even this method of average calculation of individual groups of weight only when accurate figures cannot be supplied.

(2) Under *direct or productive wages* are understood the wages paid for moulding, core-making, cleaning and dressing of the castings, even if they are not paid to the skilled men, but partly to their assistants for actual help on these activities. It is not correct to consider the wages for cleaning and dressing as a part of the production overheads as they are dependent on the form of the castings, and must be paid in order to change the castings into saleable goods. As they are to some extent proportional to the wages for moulding and core-making, the wages for cleaning and dressing can be taken into account for the purposes of estimation, by a percentage addition to the moulding and core-making wages.

(3) *Production overheads* are (*a*) all materials not included in the molten iron, e.g., sand, coal dust, graphite, charcoal, coke for drying stoves, chaplets, core-irons, oils and fats, parting sand, fire-wood, shovels, riddles, files, ladles, grinding wheels, etc. (*b*) All wages not already taken into account as direct wages. (*c*) Other overheads, such as salaries of staff members engaged in production work, including contribution to insurances, etc., transport costs, e.g., horses, carts, lorries, depreciation of production equipment, tools and repairs ; consumption of power, as far as not included in the costs of molten iron ; heating and lighting of shops and offices belonging to production ; maintenance of general patterns not covered by special invoice to the customer ; costs of storage ; employers' withdrawals, in so far as they can be considered as equal to a salary of a managing director or general manager. (It is usual in some foreign countries, in contrast to British custom, to include such a position in the costs in order to put private business and limited companies, etc., on the same footing for comparison purposes. It is a matter of agreement whether to accept this or that point of view.)

(4) *Selling and distribution costs are* salaries of staff members engaged in selling and distributing the produced goods ; insurances against breakage, fire, accidents, etc. ; travelling expenses ; commissions ; advertisement ; interests ; rents and taxes ; stationery and other office needs ; welfare expenditure.

It may be added that the percentages given for the standard calculation, i.e., 200%, 18% and 12%, need a continuous supervision. It rests completely with the works concerned whether they think it advisable to alter them, considering the special circumstances under which they have to work. For example, in an article on the subject in *Die Giesserei*, of May, 1941, the percentage of 200% is changed to 120% for green-sand casting, or 150% for dry-sand casting.

The writer of this report would draw from his experience the conclusion that 200% is rather high, but this may be the sequence of the assumption that for the calculation of standard costs the working conditions should be taken as a basis when the foundry is working only 40% to 60% of its capacity. It is certainly correct not to use 100% activity as basis, but to have some reserve for unforeseen incidents, machine trouble, unexpected change of programme, urgent deliveries ; but as it is well known that the costs per unit of production decrease with an increasing degree of activity, it would be dangerous to enlarge this reserve too much. Rather is it recommended to take as a basis for the calculation of standard costs 80% to 90% of the capacity of the shops, after this capacity has been found by careful investigations, based on statistics and work and time studies.

Combined Aluminium Committee

A COMBINED ALUMINIUM COMMITTEE, representing the United Kingdom, the United States and Canada, has been established to co-ordinate the activities of the three countries in respect of aluminium. Charles E. Wilson, Executive Vice-President of the War Production Board in Washington, will serve as chairman. The other members are Sir Richard Fairey, Director-General of the British Aircraft Commission in Washington, and George C. Bateman, Metals Controller of Canada.

The Committee, which is the fourth of its kind to be appointed, will report its findings and actions to the Combined Production and Resources Board and the Combined Raw Materials Board. A Technical Sub-Committee, consisting of Arthur H. Bunker, Director of the War Production Board's aluminium and magnesium division, and P. W. Rolleston, Director of Raw Materials and Supply, Supply Services, British Aircraft Commission, has been appointed to prepare detailed estimates of 1943 and 1944 production and requirements of the three countries.

The first of the special Committees was appointed on December 15 to study the steel situation. A Committee on Copper was appointed on February 10, and a Committee on Rubber was appointed on February 16.

The Surface Protection of Magnesium Alloys

An investigation has shown that films possessing excellent corrosion-resisting properties can be formed on the binary magnesium-manganese alloy AM 503 by electrolytic treatment in a hot, dilute solution of chromic acid, using either a.c. or, preferably, a.c. and superimposed d.c., adjusted so that the mean value of the applied e.m.f. is negative. The work is reported by Parkinson and Cuthbertson, from which the present abstract is taken.*

WITH comparatively few exceptions magnesium alloys intended for use in the open must be protected against corrosive influences if they are to give a useful service life. A considerable amount of work has been done to improve the corrosion-resisting properties of these materials by some form of surface protection treatment, and the methods developed are directed primarily towards the production of metallic or non-metallic coatings. In practice the latter has so far proved the more important, particularly those coatings produced by anodic treatment, dipping methods, and by paints or lacquers.

At the present stage of development experience indicates that, in general, the best results are to be obtained by resorting to a duplex process, involving either a chemical or electrochemical pre-treatment followed by spray-painting. The steady increase in the demand for magnesium alloys and the ever-widening sphere of their application to-day are creating a steady demand for better protection against corrosion, and it is for this reason that an investigation was undertaken by Parkinson and Cuthbertson, and much data is given in their report of the work and its results.

The investigation was preceded by a preliminary survey of the behaviour of metallic magnesium under electrolytic treatment in a wide variety of aqueous electrolytes. The protection of the two commercial alloys, AM 503 and A 8, next received consideration, and after some further exploratory work a process involving treatment in an aqueous solution of chromic acid, using either alternating current or, alternatively, direct current superimposed on alternating current, was developed and studied in some detail. The analyses of the magnesium and alloys are reported as in Table I.

TABLE I.—ANALYSES OF MATERIALS.

Material.	Aluminium, %.	Zinc, %.	Silicon, %.	Copper, %.	Manganese, %.	Iron, %.	Magnesium, %.
Magnesium	—	—	—	—	—	—	99·99
AM503 (rolled)* ...	—	0·2 max.	0·4 max.	—	2·0	0·02 max.	difference.
A8 (cast)	8·5	0·48	0·08	0·12	0·22	0·03	difference

*Nominal composition.

The degree of protection obtained under the various conditions investigated was estimated by means of corrosion tests. Three methods of testing were adopted—continuous-immersion tests, usually of two to three weeks' duration, at room temperature; intermittent immersion tests, in which specimens were hand-dipped, usually three times a day, for approximately 15 secs., 60 dips constituting a complete test; salt-spray test, in which specimens were suspended inside a glass box and constantly bathed in a fine mist formed by compressed-air atomisation of the corroding medium; the duration of this test was two or three weeks. The behaviour of the specimens under test was estimated by visual examination, frequently supplemented by loss of weight determinations.

Painted specimens included in some of the corrosion tests were prepared as follows: After prefilming, the samples were given a full sprayed coat of zinc-chromate pigmented primer (Cellon C 933); a week later a sprayed coat of grey cellulose enamel (Cellon C 600) was applied, being followed, after the lapse of a further two days, by a second coat. The specimens were then allowed to stand for two days before finally being subjected to the corrosion tests.

In the investigation of electrolytic methods of protection preliminary tests of an exploratory character were conducted, using magnesium electrodes subjected both to anodic and to alternating current (50 cycle/sec.) treatment in a variety of electrolytes. Films of variable protection value were produced, but in no case was it found possible to build up thick films such as can be obtained by the anodisation of aluminium; nor was any film comparable with the latter from the protective viewpoint. As alternating current rather than direct current electrolysis appeared to offer the more promising prospects, further alternating current tests were instituted, in which magnesum was treated in a selection of electrolytes, the concentration of baths being 3%, unless otherwise stated, and the working temperature approximately 20° C. The results of these tests are summarised thus:—

(*a*) Solutions containing, respectively, sodium borate, phosphate, oxalate, or acetate, oxalic acid, and chrome alum produced either no surface film, or at the best an extremely thin deposit affording little or no protection. When immersed in sea-water, the treated specimens showed evidence of hydroxide formation after the lapse of about 3 hours.

(*b*) Solutions containing potassium fluoride gave white films having a fairly high electrical resistance, but which broke down suddenly at about 25 v. Additions of 0·5% sodium hydroxide to this solution improved the insulating properties of the films, enabling them to withstand 75 v. without rupture. The films obtained from each of these solutions, however, were no more resistant to corrosion than were those mentioned in (*a*) above.

(*c*) A 1% solution of sodium hydroxide gave white films, similar to those produced from fluoride baths, but slightly more resistant to corrosion; the insulating properties were inferior, however, and the breakdown voltage lower, viz., 15 v.

(*d*) A solution of sodium aluminate was chosen in an endeavour to deposit a film of alumina. The results were disappointing, and on the whole resembled those obtained from sodium-hydroxide solutions.

(*e*) Solutions containing silicates, and mixtures of silicates with fluorides, hydroxides, and carbonates in every case produced white films affording little or no protection.

(*f*) Chromic acid at fairly low current densities gave white films which provided protection against corrosion in sea-water for 72 hours.

Apart from chromic acid no other acid solution has been found capable of forming a coherent film on magnesium, and its potentialities were investigated in greater detail.

Tests were carried out on alternating current treatment of the magnesium alloys AM 503 and A 8 in chromic acid. The concentration of the bath used was increased from 3 to 5%, there being indications that somewhat more

* N. Parkinson and J. W. Cuthbertson. *Journ. Inst. Metals*, March, 1943, pp. 109–130.

resistant films were thereby obtained. The electrolyte was contained in a glass cell having a capacity of 2 litres, provided with air agitation and heated and maintained at a constant temperature by an oil bath.

Alloy AM 503.—Many samples were tested at nominal current densities of 0·5—3·0 amp./dm.² and temperature of 20°—90° C., the period of treatment being 10 mins. Particulars of some of these tests, together with data on the appearance of the films obtained, are included in Table II. It will be noted that an increase in temperature is accompanied by a progressive darkening of the film, the colour changing from light grey or brown almost to black. At any constant temperature an increase in current density also darkens the film, but in this case the extent of the darkening appears to depend on the actual temperature, becoming more pronounced as the latter is raised. With the exception of those formed at temperatures exceeding 80° C., the darker films almost invariably proved to be the most resistant to corrosion. When the electrolyte is above 80° C. there is some tendency towards local pitting, and evidence of a general deterioration in the corrosion-resisting properties. A high-current density appears to be desirable; above 2·5 amp./dm.², however, throwing-power effects become increasingly evident, the action of the current being confined more and more to the side of the specimen directly facing the stainless steel electrode. In practice, by a suitable disposition of the electrodes, this difficulty probably could be overcome.

TABLE II.—A.C. TREATMENT OF AM 503 IN 5% CHROMIC ACID.

No. of Specimen.	Current Density, amp./dm.²	Temp., °C.	Colour of Film.	Remarks.
1	1·87	20	Very light brown	
2	1·25	40	Light grey	
3	1·25	40	"	Gas evolved. No capacity effect. Side facing steel electrode darker in colour.
4	1·87	40	Grey	
5	1·87	40	"	
6	0·62	40	Light grey	
7	0·62	40	"	
8	1·87	60	Grey	
9	1·87	60	"	
10	2·50	60	Darker grey	Greater evolution of gas.
11	2·50	60	"	Small capacity effect persisting for 3 mins.
12	3·12	60	Dark grey	
13	3·12	60	"	
14	0·62	60	Light Grey	
15	0·62	60	"	Gas evolved initially. Side facing steel electrode darker in colour.
16	2·50	30	Transparent brown	
17	2·50	30	" "	
18	1·25	30	" "	
19	1·25	30	" "	
20	0·62	30	" "	
21	0·62	30	" "	
22	0·62	50	Transparent brown	
23	0·62	50	" "	Initial evolution of gas.
24	1·25	50	Light grey	
25	1·25	50	"	
26	2·5	50	Darker grey	Gas evolved. Side facing steel electrode darker in colour.
27	2·5	50	Dark and light grey	
28	2·5	70	" "	
29	2·5	70	" "	
30	1·25	70	" "	
31	1·25	70	" "	
32	0·62	70	Light grey	
33	0·62	70	"	
34	0·62	80	Light brown	
35	0·62	80	"	Evolution of gas increasing as the higher temperatures are approached.
36	1·25	80	Darker brown	
37	1·25	80	"	
38	2·5	80	Dark grey	
39	2·5	80	"	
40	2·5	85	Almost black	
41	2·5	85	"	
42	1·25	85	Dark grey	
43	1·25	85	"	
44	0·62	85	Cream-coloured	
45	0·62	85	"	

The corrosion-resisting properties of the specimens listed in Table II were estimated qualitatively from their appearance after subjection to the salt spray for 400 hours, or quantitatively by determining the loss in weight under intermittent immersion. Comparison specimens protected according to the methods patented by Sutton and his collaborators* were included in the salt-spray test. Under this latter test all the specimens proved to be inferior to those treated by the R.A.E. 6-hour method. On the other hand, with certain exceptions, specimens were superior to those chromated in the R.A.E. 30-minute bath. Figures for loss of weight are given in Table III, which show the advantages of operating the cell at a high temperature and current density.

TABLE III.—INTERMITTENT DIPPING TESTS ON AM 503 TREATED WITH A.C.

No. Specimen.	Loss in Weight after 60 Dips in 10% NaCl, grm./dm.²	No. Specimen.	Loss in Weight after 60 Dips in 10% NaCl, grm./dm.²
1	0·383	25	0·137
3	0·244	27	0·105
5	0·168	29	0·088
7	0·293	31	0·096
9	0·105	33	0·120
11	0·082	35	0·110
13	0·087	37	0·087
15	0·127	39	0·090
17	0·138	41	0·099
19	0·166	43	0·097
21	0·496	45	0·124
23	0·276	—	—

The foregoing technique was modified in an attempt to effect a further improvement. Instead of the AM 503 stainless steel combination, two similar AM 503 electrodes were substituted, while the duration of the treatment was extended from 10 to 30 mins. The same electrolyte was used, the temperature and current density being maintained at 70° C. and 2·0 amp./dm.², respectively. Dark-coloured films were produced which behaved well under the salt spray, the average loss in weight being below the average recorded for the chromated comparison specimens (Table IV); the maximum and minimum weight losses for the alternating current treated specimens are each approximately 30% lower than the corresponding losses for the specimens treated by the R.A.E. 6-hour process—under these corrosive conditions apparently much the more protective of the two R.A.E. treatments.

TABLE IV.—SALT-SPRAY TESTS ON AM 503 TREATED WITH A.C. 400 HOURS.

A.C. Treatment.		R.A.E. 6-hour Bath.		R.A.E. 30-minute Bath.	
No. of Specimen.	Loss in Weight, grm./dm.²	No. of Specimen.	Loss in Weight, grm./dm.²	No. of Specimen.	Loss in Weight, grm./dm.²
46	0·0541	A1	0·0653	B1	0·574
47	0·0322	A2	0·0721	B2	0·816
48	0·0210	A3	0·0470	B3	0·521
49	0·0563	A4	0·0321	B4	0·932
50	0·0371	A5	0·0952	B5	0·376
51	0·0652	A6	0·0781	B6	0·663
Average weight loss	0·0443	Average weight loss	0·0649	Average weight loss	0·647

Alloy A 8.—Attempts to form a protective film on this alloy by the above method, despite wide variations made both in the current density and temperature, were unsuccessful. At low current densities and/or low temperatures, light-coloured non-protective films are formed; on the other hand, on treatment for 1 hour at a current density of 6 amp./dm.² and temperature of 70° C., an extremely thin black film was obtained, but again its protective value proved to be practically nil.

A.C. plus D.C. Treatment.—Compared with films produced by anodic oxidation, those formed by alternating current electrolys isusually are thinner. Superimposition of direct current suggests itself as a possible means of improving the alternating current process in this respect, and of so producing even better films. A circuit was therefore arranged and provision made to supply the cell simultaneously with 0—7 v. a.c. from a transformer, and with with 0—12 v. d.c. from an accumulator. The same cell was used as before, but it accommodated three electrodes, two of the alloys undergoing treatment and an intermediate plate of stainless steel. The electrolyte was a 5% solution of chromic acid, and was maintained at 70° C.

* Amongst other methods of protection the two chromating processes known as the R.A.E. 6-hour[1] and R.A.E. 30-mins.[2] processes, respectively, have become very popular. The former involves treatment in a gently boiling solution containing potassium dichromate, 1·5%; potash alum, 1·0%; and sodium hydroxide, 0·5%; the latter treatment in a briskly boiling solution containing ammonium sulphate, 1·5%; potassium dichromate, 1·5%; and ammonium hydroxide ($d = 0·880$), 0·4%.

1 British Patent No. 331,853.
2 British Patent No. 353,415.

Specimens of AM 503 treated by the combined process, using a current density of 2·5 amp./dm.² (a.c.) + 0·5 amp./dm.² (cathodic), along with comparison samples filmed respectively in the R.A.E. 6-hour, R.A.E. 30-minute, and acidchromate baths, were subjected to corrosion tests. The weight losses incurred, given in Tables V and VI, indicate that under intermittent corrosion the new treatment occupies a position intermediate between the R.A.E. 6-hour and 30-minute treatments, whereas under the salt-spray it is superior to the other three methods of protection. By comparison with the figures for salt-spray tests previously given in Table IV, it is obvious that the combined process affords improved protection against this type of attack.

TABLE V.—INTERMITTENT DIPPING TESTS ON AM 503 TREATED WITH A.C. + D.C. 60 DIPS.

Electrolytic Process.		R.A.E. 6-hour Bath.		R.A.E. 30-minute Bath.		Acid Chromate Bath.	
No. of Specimen.	Loss in Weight, grm./dm.²	No. of Specimen.	Loss in Weight, grm./dm.²	No. of Specimen.	Loss in Weight, grm./dm.²	No. of Specimen.	Loss in Weight, grm./dm.²
P13	0·0091	S13	0·0075	R13	0·0793	IG13	0·1005
P14	0·0250	S14	0·0087	R14	0·0443	IG14	0·0548
P15	0·0352	S15	0·0076	R15	0·0420	IG15	0·0715
P16	0·0400	S16	0·0113	R16	0·0239	IG16	0·0954
P17	0·0274	S17	0·0074	R17	0·0615	IG17	0·0661
P18	0·0285	S18	0·0092	R18	0·0739	IG18	0·0463
P19	0·0491	S19	0·0096	R19	0·0563	IG19	0·0709
P20	0·0309	S20	0·0080	R20	0·0470	IG20	0·0175
P21	0·0245	S21	0·0114	R21	0·0377	IG21	0·0947
P22	0·0332	S22	0·0060	R22	0·0170	IG22	0·0885
P23	0·0379	S23	0·0077	R23	0·0201	IG23	0·0232
P24	0·0451	S24	0·0065	R24	0·0305	IG24	0·0921
Average weight loss	0·0322	Average weight loss	0·0084	Average weight loss	0·0445	Average weight loss	0·0601

A number of corrosion tests on painted specimens of AM 503, mainly qualitative, were completed. In one of these, four series, each comprising 12 specimens, treated respectively by the combined electrolyte, R.A.E. 6-hour, R.A.E. 30-minute, and acid-chromate processes, were painted according to the directions previously outlined, and were then immersed continuously for 700 hours in sodium-chloride solution. On removal none of the specimens was visibly corroded, and from examination after stripping it was concluded that all the pretreatments had proved serviceable.

TABLE VI.—SALT-SPRAY TESTS ON AM 503 TREATED WITH A.C. + D.C. 400 HOURS.

Electrolytic Process.		R.A.E. 6-hour Bath.		R.A.E. 30-minute Bath.		Acid Chromate Bath.	
No. of Specimen.	Loss in Weight, grm./dm.²	No. of Specimen.	Loss in Weight, grm./dm.²	No. of Specimen.	Loss in Weight, grm./dm.²	No. of Specimen.	Loss in Weight, grm./dm.²
P1	0·0307	S1	0·0354	R1*	0·6397	IG1	0·4736
P2	0·0551	S2	0·0504	R2	0·5116	IG2	0·5866
P3	0·0307	S3	0·0330	R3	0·6123	IG3	1·0310
P4	0·0361	S4	0·0432	R4	1·1582	IG4	0·6626
P5	0·0220	S5	0·0407	R5	0·6431	IG5	0·9740
P6	0·0229	S6	0·0909	R6	0·8239	IG6	0·6880
P7	0·0254	S7	0·1067	R7	0·6002	IG7	0·4517
P8	0·0342	S8	0·1869	R8	0·3722	IG8	0·7664
P9	0·0205	S9	0·0749	R9	0·4960	IG9	0·5172
P10	0·0224	S10	0·1190	R10	0·3902	IG10	0·5543
P11	0·0207	S11	0·1655	R11	0·5470	IG11	0·6462
P12	0·0285	S12	0·1024	R12	0·4664	IG12	0·5113
Average weight loss	0·0291	Average weight loss	0·0781	Average weight loss	0·6651	Average weight loss	0·6553

The use of superimposed d.c. effected no discernible improvement in the poor results previously mentioned when considering the treatment of A 8 by a.c. alone. When subjected to a.c. and simultaneously made cathodic, negative results were obtained; on the other hand, when made anodic to the extent of 2·0 amp./d.c.², the alloy developed a dense white film which, however, proved to be non-protective, as immersion in salt solution led to incipient corrosion after some 6 hours.

Fluorescent Magnetic Inspection

A new magnetic inspection method is described which appears to be capable of speeding up visual examination and avoids many of the troubles due to false indications and high lights.

ALL inspection methods depend to a greater or less extent on visual examination. The use of rules, dial gauges, micrometers and other measuring devices require the reading of a scale, magnetic analysis inspection relies on the reading of a voltmeter, and X-ray and gamma-ray radiography depend on the interpretation of the lights and shadows of the films. In a similar way, the magnetic particles method is carried out either with Magnaflux dry powders or with pastes for the wet method requires careful examination of the accumulations of the particles that form the indications of important discontinuities. The description by W. E. Thomas* of a new magnetic inspection method and the comparisons of its usefulness with the standard Magnaflux test is therefore of interest to all users of metals.

From the purely visual examination standpoint, magnetic particle inspection requires much care in carrying out, as there are a number of factors contributing to its success. A careful scanning of the entire surface is necessary if indications of discontinuities are not to be overlooked, as some of these are very small in size. Masking or partial masking of indications by the colour of the background, irregular surface finish, holes or machines grooves, false indications and the relatively small volume of particles over sub-surface discontinuities also occurs. Parts inspected by the wet method are covered with a light distillate oil used as a suspension medium for the paste particles, and this oil film causes disturbing high-lights. Highly finished parts examined by either the wet or dry methods also cause high-lights which interfere with visual examination.

Because of these factors and because of the widely increased use of this method of testing, investigations have been carried out to improve its sensitivity and visibility. The original grey powder used has been replaced by black and red powders, and pastes for the wet method have also been developed in black and in red. Recent investigations have resulted in the development of fluorescent indicating particles produced under the name Magnaglo, suitable for the wet method of testing. A fluorescent dry powder is in the development stage.

In making up a wet bath for the magnetic particle method, the suspension commonly used as a 1 to 1·5% solid particles in a light oil medium. The amount of paste is as large as possible, but small enough to avoid an objectionable overall coating of the part by the paste. Even a 1 or 1·5% suspension leaves a certain amount of particles all over the part, but their extremely small size makes the film practically invisible. Under black light, the term applied to invisible radiant energy in that portion of the ultra-violet spectrum just beyond the blue of the visible spectrum, and having a range between 3,000 and 4,000 Avigstrom units in wave length, fluorescent magnetic particles are much more visible than Magnaflux paste particles of the same size under visible light. Therefore the same 1 to 1·5 suspension of fluorescent paste leaves a very visible and disconcerting fluorescent film over the part being examined. It is necessary to reduce the per-

* *Metals and Alloys*, 1942, vol. 16, No. 5, pp. 886–890.

centage in suspension drastically to eliminate this film, so that the percentage actually used ranges from 0·2 to 0·3% by volume of paste to volume of oil. When this small amount of paste is used the indications of discontinuities while readily visible under black light are not easily seen under visible light, because the volume of the build-up of the paste particles is very small. If this is unsuitable for inspection purposes a mixture of the non-fluorescent red paste and the fluorescent material may be used.

The detrimental factors which occur when using ordinary magnetic particle inspection are considerably changed when using the fluorescent paste. If a piece is examined in darkness, any small light source, such as a fluorescent indication, attracts and draws the eye immediately, and the size and complexity of the part does not, therefore, offer merely the opportunity to miss the indication as in visible light. The extreme fluorescence of the indication also makes the visibility under black light so high that even very small indications are noted. Masking of the indication is eliminated as visibility depends upon reflective values which can be accurately measured. Contrast between a very dark surface and a fluorescent indication is much greater than when using black or red pastes. Surface finish also does not disturb inspection, as it is necessary to look only for a brilliant line or indication, and not for the contrast between the indication and the surface of the part. As the indication forms its own visible rays the detection of holes or machine grooves is much easier, as it is difficult to project a visible light into such discontinuities and still have the eye in a favourable position to view the indication. False indications, particularly those caused by leakage fields at sharp corners and changes of section are much reduced by the use of the fluorescent method.

Sub-surface discontinuities cause weaker external leakage magnetic fields than surface defects, because of the much greater permeability of the iron or steel over the discontinuity. These leakage fields extend over a much larger area than those caused by a surface crack. For these reasons, therefore, the extremely small ferro-magnetic particles in paste do not build up well into sharp, clearly defined heavy indications. Many indications are actually formed with those pastes which are not located by inspection purely because they appear the same as many other areas of the surface of the part which have a light visible accumulation of particles. The fluorescent particle indications, on the other hand, can be seen under black light when the same non-fluorescent particle indication is not noticeable in visible light. The question of visible light high-lights also do not seriously enter into inspection under black light as high-lights or reflections from the part are all of the near ultra-violet or invisible range.

In general, the fluorescent method of inspection appears to be capable of speeding up visual examination, which is the most time-consuming part of magnetic particle inspection, and at the same time avoids many of the troubles due to false indications and high-lights, and eliminates difficulties of colour contrast and inspection in areas difficult to view.

Mr. C. A. Masterman, M.A., F.I.C., M.I.Mech.E., M.Inst.Gas.E., has been appointed Convener of the Post-war Planning Committee of the British Gas Federation. Mr. Masterman was for many years Chief Technical Officer at the Gas Light and Coke Co.'s research laboratories, and on the outbreak of war he returned to his former work with the Chemical Defence Research, first as Assistant Director at the Ministry of Supply, and later as Acting Superintendent at the experimental station. Last year he transferred on to the technical staff of the Gas Directorate.

The gas industry has realised that the problems of reconstruction after the war will open up many phases and plans are being laid for internal reconstruction, and technical research of all kinds is being carried out.

A Rapid Method for the Determination of Zinc in Aluminium Alloys with the Simultaneous Determination of Copper*

E. I. Nikitina.

(*All-Union Institute for Aircraft Materials*).

As an alternative to the well-known method for determining zinc in aluminium alloys by electrolysis from an alkaline solution, a method has been evolved, based on that described by Lundell and Nai Kim Bee,[1] in which zinc is precipitated as the double sulphocyanate $ZnHg(CNS)_4$. The method also permits the simultaneous determination of copper in the same sample of alloy.

Previous investigations[2] had already shown that aluminium, magnesium and manganese do not interfere with the estimation of zinc, and new work has now proved that the effect of iron can be completely eliminated by addition to the solution of sodium pyrophosphate, which removes the iron as a complex without affecting the determination of zinc.

Procedure

0·2–0·5 grm. of alloy is dissolved in 25 c.c. of 10% sulphuric acid. The residue is filtered off and washed with hot water. Electrolytic and microchemical tests on the filtrate have shown that the copper present in the aluminium alloy remains entirely in this residue, provided filtration is carried out within an hour of the alloys being dissolved.

Determination of Zinc.—The filtrate is evaporated down to 15–20 c.c., and then there are added to it 15 c.c. of 10% sulphuric acid, 30–35 c.c. of 5% sodium pyrophosphate solution, and 15–20 c.c. of mercury sulphocyanate reagent. According to Lundell and Nai Kim Bee (*loc. cit.*), this reagent is made by dissolving 39 grm. of potassium sulphocyanate in 200 c.c. of water and then stirring in 27 grm. of powdered mercuric chloride as the solution is slowly diluted to 1,000 c.c. If a slight residue remains the solution is filtered.

The white precipitate of zinc mercury sulphocyanate is allowed to stand for one hour, with occasional stirring, before being collected in a weighed Gooch crucible. It is then washed five or six times with cold water containing 20 c.c. of reagent per litre, well dried, heated at 105°–108° C. in a hot oven, and subsequently weighed. The temperature of 108° should not be exceeded, as the precipitate begins to decompose at approximately 110° C. The conversion factor for zinc = 0·1312.

Determination of Copper.—The residue remaining after the alloy has been dissolved in sulphuric acid is dissolved on the filter paper in 1 : 1 nitric acid; then 1 c.c. of concentrated sulphuric acid is added, and the solution, heated to 60° C., is electrolysed in the usual way.

The method described has been tested for a series of aluminium alloys containing 0·5–3% zinc, up to 0·5% iron, up to 8% magnesium, up to 4% copper, up to 0·5% manganese, and up to 0·5% silicon. Some of the results are given in the table :—

Weight of Sample, Grm.	Actual Content, %		Determined, %		Weight of Proportion of $ZnHg(CNS)_4$, Grm.
	Zn.	Cu.	Zn.	Cu.	
0·1458	2·5	—	2·60	—	0·02897
0·1004	2·5	—	2·57	—	0·01973
0·2100	3·30	3·60	3·37	3·60	0·0537
0·4144	3·30	3·60	3·36	3·66	0·1064
0·5	0·48	4·74	0·50	4·74	0·0192
0·5	0·48	4·74	0·49	4·72	0·0188
0·5	0·48	4·74	0·51	4·73	0·0197

Mr. Francis W. Rowe has joined the Board of Kryn and Lahy (1928) Ltd., of Letchworth, as managing director. Until recently, Mr. Rowe was on the Board of David Brown and Sons (Hudds.), Ltd., and was general manager of their Penistone Works; he had been associated with this firm for nearly 20 years.

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* *Zavodskaya Laboratoriya*, 1939, **8** (12), 1314–1315.

1 G. E. F. Lundell and Nai Kim Bee. *Trans. Amer. Inst. Metals*, 1914, **8**, 146–150.

2 E. I. Nikitina. *Zavod. Lab.*, 1938, **7** (2), 162–166.

Prolongation of Metal Patents

By S. T. Madeley

In many instances war conditions seriously interfere with the successful exploitation of patents, and, due to this cause, the remuneration patentees would receive under normal conditions is inadequate. While it is known that a patentee may apply for an extension this brief review of important judgments may guide those seeking extensions for their patents.

ATTENTION has recently been drawn to the importance of patents relating to metal by the issue of the report of the Committee on the Development of Hydro-electric Power in Scotland. Proposals have been made by the Committee for very considerable electro-metallurgical plant in Northern Scotland, where water-power is best available. Aluminium, magnesium, and ferro-alloys are amongst the metals dealt with. If this report is followed up by practical steps, and it is devoutly to be hoped that this is the case, no doubt matters will be soon pushed on after the war as one factor of our recovery.

There must be many important patents relating to the metal industries which are now about to expire and which have failed owing to the war to remunerate the patentees in an adequate manner. It is well known that Section 18 (6) (7) of the Patents and Designs Acts as amended by the 1942 Act allows a patentee to apply for an extension of term not exceeding 10 years on the sole ground of loss or damage due to hostilities. Furthermore, it is appreciated that Statutory Rules and Orders, 1941, No. 1780, and 1942, No. 1882, give the Government exceedingly wide powers over inventions and patents in this country. Also, such rights and interests in the U.S.A. regarding these forms of industrial property as may be in the present or future possession of residents of the United Kingdom are also under the control of the Government. It will be instructive to review the more important judgments since the 1942 Amending Act came into force, and to see how they affect the situation.

Although no metallurgical patent has yet, so far as the writer is aware, come up for decision under Section 18 (6), we get a not very remote case in the Aiton patent, which relates to iron pipes. There was a parent patent No. 273557, and a patent of addition, No. 291656. They relate to methods and means for making pipe joints and were due to expire on November 3, 1942. The registered owners of the patents were Aiton and Co., Ltd., and Sir John Arthur Aiton, who applied by Originating Summons for extension of the patents under Section 18 (6) (7). In business as pipe-work engineers, the company manufactured and installed large quantities of wrought iron, cast iron, and steel pipes. New and improved methods of fastening joints in wrought-iron steel pipes for use in high-pressure electrical work formed the subject matter of the patents. As soon as the patents had been lodged, the company commercially exploited the inventions.

The peak years were 1931, 1935, 1937. Slump conditions adversely affected the business in 1932, 1933, and 1934. Fear of war, followed by the outbreak of hostilities, caused a progressively accelerated fall in output. Most of the joints were made under the patent of addition. Royalties from licencees and fees from foreign patents were small. Utillty was deposed.

Objection was raised as to whether there had been actual user of the parent patent. On this point Mr. Justice Simonds took a lenient view. He decided that some loss owing to the war had been proved. Substantial user despite the war did not in his view justify prolongation for the whole war period up to the date of the Hearing. Taking into account only the loss up to that date (November 3, 1942), making no provision for any further loss which might be incurred if the war were prolonged, or which might occur by reason of dislocation and difficulty of exploitation after the war has ended, the judge prolonged the patents for 18 months from November 3.

The importance of the bearing of foreign patents on prolongations under Section 18 (6) was early emphasised by the same judge when he heard an application which was made by Originating Summons by Firmosec, Ltd., and Mr. J. Duffy, for extensions of their Letters Patent No. 260906.

In this case the Judge said :—

> " It is incumbent in a case where the patentee relies on sub-section (6) of Section 8, to give such information as he can to the Court in regard to any profits which have been made in respect of foreign Letters Patent, because it might be that what he has lost by reason of the war in respect of trade in this country he has gained by trading in a foreign country, where he is covered by his foreign patent. In this case I am satisfied that there was no such profit to be brought into account against loss incurred in this country. I limit the prolongation to a period of two years and eight months. Under the Amending Act of 1942, Section 1, it will be open to the patentee to ask for a period of further prolongation, and that may be based either upon a further continuance of the war or upon the fact that by reason of a difficulty in obtaining suitable seasoned material he is unable to exploit his patent for a period after hostilities have ceased."

Where there has been failure to supply the fullest information available about receipts from foreign patents and kindred forms of property, the extension granted has been very much reduced.

Somewhat different circumstances ruled the situation as regards application by Radiation, Ltd., in respect of their patent No. 261609, which had lapsed when they applied by Originating Summons for an extension under Section 18 (6). Sales increased up to and including 1936, but decreased thereafter and had a catastrophic fall in 1939. This decline was not due to the saturation of the market with patented articles, but to the international conditions, which militated against a vigorous policy regarding sales propaganda. advertising, and so on, by those interested in the patent. Resources were to some extent diverted to produce war material. Since the outbreak of war there had been limitation and control in the national interest by the Government. Essential raw materials were restricted. Labour was short. Available manufacturing sources of the company had been diverted to war supplies. Expenditure was restricted. Fall-off in sales has not been off-set by a corresponding fall in overhead. There was always a lag. Mr. Justice Simonds regranted the patent for two and a half years.

Actual loss through destruction of stocks and works by enemy action was a factor in the extension applied for in the case of the Brackensey Patent No. 270571. Most of the business done by the Applicant Company, Franco-British Electric Coy., Ltd., lay in contracts to provide space for and erect and maintain the patented article for fixed periods at a weekly or monthly fixed sum. There was no prospect of an increased post-war demand making up for war loss, which would very largely destroy the business goodwill in default of prolongation. An extension of three

years and two months—i.e., the full period of war loss was granted. This was without prejudice to a further application.

Now for a last case, in which a series of patents had been assigned to the Crown under Sections 29 and 30 of the Patents, etc., Act (which relate to Crown user and secrecy). These patents were admittedly of great merit. They were the Meredith patents Ncd. 365187, 365187, and the Cooke patents 365189 and 365190. After six years' user the Crown reassigned them to the inventors under an agreement whereby the President of the Air Council received 45% of the proceeds of commercial exploitation and was able to revise the terms of that agreement. An exclusive licence had been granted. Some £5,000 was made by the Petitioners out of their British patents and another £5,000 or so out of their foreign. The licencees had practically neither gained nor lost. Eight years' regrants were made in each case, in spite of objection by the Comptroller-General.

The following is the gist of the Judge's rulings :—

The Crown need not assess the value to it of inventions assigned to it during the period of assignment. Adequacy of payment offered or made to a service inventor for his inventions will not be considered by the Court. When such reward is covered by Treasury Regulations, assignment of a patent for a period to the Crown under Section 30 of the Acts cannot be relied on by a Petitioner as having deprived him of a chance of exploiting the patent during such period.

When a specified Secretary of State can alter the terms of an agreement between the Crown and an inventor relating to the division of the proceeds of commercial exploitation of the invention, the terms will not be reviewed by the Court.

The position of an ordinary employee who has contracted concerning inventions he makes during his employment should be the standpoint for that of considering a service inventor in prolongation proceedings.

From the above brief résumés of judgments it appears that substantial merit is necessary to obtain a good extension. In the ordinary way, under Section 18 (6) (7) the maximum period of prolongation granted will equal that between the beginning of the war and the expiration of the patent, but further extensions may be applied for.

Recent Improvements in Stopping Blast Furnace Tap-Holes

By A. G. Arend

Improved mechanism for the stopping of blast-furnace tap-holes is discussed. Attention is directed to its automatic adjustment, to its efficiency in coping with difficulties encountered in this operation, and to the reduction in risk to operators by its use.

MANY existing methods of stopping the tap-holes of blast-furnaces are fraught with a certain amount of danger to the operatives, offer practical difficulties where plugging material has to be repeatedly applied, and cannot efficiently cope with tap-holes where the surrounding brickwork has been seriously eroded. An improved system of working makes it possible to successfully meet these different requirements, apart from automatically bringing the machine to the desired position and adjusting it, without necessitating the usual manual attention.

The type of mechanism recently introduced makes it possible to move the unit into position in front of the tap-hole by means of what is termed a locating outfit. This consists of an arrangement of jib-crane equipped with parallel motion guides in both vertical and horizontal planes, wherein a slewing cylinder turns the jib, thus positively guiding the machine by horizontal parallel rod linkage, until it has reached the desired position in front of the tap-hole. Before the actual working position has been attained, the action of a lever causes the shaft of the jib to rotate, and which is guided along a cam so that the parallel sections of the vertical members, which may be in rectangular position, are converted to oblique position. In this manner, the barrel of the machine is suitably inclined. What are known as " turn-buckles " are then used to adjust this inclination to meet the particular requirements of the slope of each tap-hole, when the nozzle of the machine is thrust into position over the shaft upon a forked lever, by the action of a cylinder. Compressed air is then used to first actuate the pressure piston, immediately followed by the control piston.

By working in this manner, the throttle-slide is raised, and the way freed for plugging material to be passed through. In other words, the working piston is moved forwards by the action of the piston in the pressure cylinder, and the material is forced into the hole with uniform precision.

In most cases there is usually a sufficiency of contents in the machine to ensure a thorough closing of the hole in the one operation, but should contingencies arise where this requires to be duplicated, provision is made to meet this. A cut-off slide closes automatically after the first operation, while the piston is retracted for the placing of another charge, so that the plug is thus secured against expulsion, and which incidentally forestalls any risk of accident or injury to the attendant.

The operating stand, where the machine movements are controlled, is purposely set at some distance from the tap-hole, so that again the risk of individual injury is minimised.

The idea of incorporating parallel-motion mechanism is so that any adjustments are simply performed by a hand-wheel which functions over a screw spindle, while the latest models make use of double square-threaded screws, so that no danger of undue slipping can arise. This method of working holds the additional advantage that should the neighbouring refractory material show a tendency to break, or should the customary position of the tap-hole have to be altered, the machine is put to no practical difficulty.

Automatic Duplicating of the Plugging Operation

In actual practice, it is not an infrequent occurrence to find the brickwork behind the tap-hole somewhat seriously eaten away, and it is not merely sufficient to close the hole, without attending to the former at the same time, since the erosion process is generally a progressive one. For this reason, the sudden thrusting of plugging material into the aperture has of necessity to be avoided, and in its stead a slow motion is employed whereby the material is so pushed forward that it forms a mushroom within the interior of the furnace. Experienced blast-furnace attendants are aware when the immediate vicinity of the tap-hole is likely to show the effects of erosion, and this can be confirmed by piercing suitable points. It is then

that the foregoing outfit can be made to perform the dual operation of plugging the actual tap-hole, and also preventing any undue breakaway in the neighbouring refractory, by adapting the same system of parallel linkages, which can be easily adjusted. In the ordinary way, many of the existing tap-hole stopping machines cannot be so readily manipulated, as the placing and adjustment by hand is not only tedious, but cannot successfully cope with some difficulties that arise, while the filling material is very limited in quantity and often necessitates an entirely fresh operation.

In such instances, the retraction of the piston for filling purposes affords the opportunity for the pressure of the blast-furnace to react upon the plug, which, being freshly applied, is somewhat soft and weak, and liable to be completely expelled. A dangerous aspect is the possibility of molten metal flowing into the machine through the exposed nozzle. It is then possible for the moisture of the remaining plugging material to cause serious explosions, whereby the whole mechanism might become impaired, apart from the danger to the operatives. An earlier alternative was to reduce the pressure of the blast to a fixed minimum; another alternative was to introduce machines equipped with a number of different cylinders, so as to provide an ample supply of the necessary material, but these were complicated to manipulate, and were unduly expensive. In the foregoing system, the one cylinder is used throughout, but with the mechanical locating device, as many stopping treatments can be applied as are wanted.

The success of this system is in no small measure due to the capacious funnel which accommodates the raw material, before it enters the working chamber, and the automatic stop-slide between the mouthpiece and the ram, which is automatically operated as the control piston functions.

The Electro-Deposition of Silver on Magnesium

The problem of plating magnesium with a nobler metal is largely one of a preplating treatment, according to Bowes and Gilbertson, who, as a result of experimental work, have developed a method which is described.*

THE electro-deposition of the nobler metals on active metals, such as aluminium and magnesium, from aqueous electrolytes presents a number of difficulties. A cleaned magnesium surface reacts very readily with water, displacing hydrogen. Silver is displaced by magnesium from neutral and acid solutions. These considerations impose certain restrictions upon the electro-deposition of silver upon magnesium in that an alkaline plating bath must be used. However, with a plating bath of a pH above 10·6, a film of magnesium hydroxide forms on the magnesium, preventing the adherence of the silver plate.

In the experimental work, the problem was to find a surface pre-treatment that would provide an adherent basis for plating from an alkaline bath. In the past many methods have been devised by which magnesium can be coated with such compounds as the oxide, chromate, dichromate, fluoride, or selenide. The methods involve either simple immersion or electrolysis. None of these methods is considered to be satisfactory for the plating of silver on magnesium.

Recently, there was published a method for the electro-deposition of nickel on magnesium and magnesium alloys.[1] This method involves a special preliminary treatment of the magnesium surface followed by plating in a boro-fluoride electrolyte.

The cyanide-carbonate silver-plating bath commonly used for silver plating is not suited to the electro-deposition of silver on magnesium. The deposits are loose, and the basis metal becomes badly pitted during plating. Elimination of the carbonate from this bath tends to reduce the pitting and to partially eliminate the reaction of the electrolyte with the cathode, but does not produce satisfactory silver deposits.

The magnesium metal used in the present investigation was produced by electrothermic reduction, distilled, extruded, and hot-rolled. Cathodes measuring 2·5 cm. × 4 cm. × 0·5 cm. were plated in glass cells containing 200 ml. of electrolyte. Anodes were 999 + fine silver. During the course of this work it was observed that rough cathodes pitted more readily than smooth cathodes; hence the magnesium was polished to a bright finish before treatment and plating. For polishing we used successively finer emery paper, finishing on a cloth-covered wheel with six-hour treatment with alumina and liquid soap.

Degreasing.—Vapour degreasing methods proved unsatisfactory, resulting in non-adherent silver plate. Satisfactory cleaning was accomplished by the action of alkaline baths that were employed later in the pre-plating cycle.

Preparation of the Metal Surface.—The following procedure is satisfactory for preparing the magnesium surface for silver plating:—

1. *Cathodic cleaning* in a bath of the following composition:

Sodium carbonate	56 g./L (7·5 oz./gal.)
Trisodium phosphate	110 g./L (14·7 oz./gal.)
Sodium hydroxide	28 g./L (3·7 oz./gal.)
Ammonium lauryl sulphate	2–8 g./L (0·2 to 1·0 oz./gal.)

This cleaning operation is carried out for 1 to 2 mins. at a current density of 20 amp./dm.² (216 amp./sq. ft.) at a temperature of 60° to 90° C.

2. *Cold water rinse.*

3. *Anodic treatment* in a bath of the following composition:

Trisodium phosphate (12 hydrate)	250 g./L (33·4 oz./gal.)

This anodic treatment is carried out at 60° to 80° C. using a current density of 30 to 40 amp./dm.² (275 to 375 amp./sq. ft.) for 3 to 6 mins. A grey film is formed on the surface of the magnesiun which is adherent, smooth and stable.

4. *Cold water rinse.*

5. *Copper strike* in a bath of following composition:

Copper cyanide	5 g./L (0·67 oz./gal.)
Potassium cyanide	100 g./L (13·3 oz./gal.)

This strike is at 25 amp./dm.² (270 amp./sq. ft.) current density, for 1 to 30 secs., depending on the condition of the metal surface or until the magnesium is covered with copper. The temperature is 20° to 25° C.

6. *Cold water rinse.*

7. *Plating* in a bath of the following composition:

Silver cyanide	8·7 g./L (1·1 oz./gal.)
Potassium cyanide	11·2 g./L (1·5 oz./gal.)
Boric acid	10 to 30 g./L (1·3 to 4 oz./gal.)

The anode-to-cathode area ratio was 1 : 6 because larger anode areas shortened the life of the bath and produced discoloured plate. The current density was 0·1 to 0·4 amp./dm.² (1·8 to 4·3 amp./sq. ft.) and the temperature was 20° to 25° C.

The silver electro-deposited on magnesium by this treatment was in the form of white, smooth, hard, adherent silver films. Plates which were allowed to lie exposed to the air in the laboratory for several months remained adherent and appeared the same as when plated, except for a normal amount of tarnishing. These films were found to be porous, and were not, therefore, suitable for protection of the magnesium surface.

* F. J. Bowen and L. I. Gilbertson. *Electrochem. Soc. Preprint* 81; 1 (1943).
1 W. S. Loose. *Trans. Electrochem Soc. Preprint* 81; 6 (1941).

A Review of Basic Open-Hearth Steel Practice at an Australian Plant

By R. L. Knight

Fundamentally the problems encountered in the manufacture of steel are similar in whatever country it is made, but differences in type and design of furnaces, in materials used, and in the general conditions of furnace operations necessitate modifications in practice to effect their solution. Knowledge on the many factors involved is cumulative the more those who can speak with authority contribute their experience to the common cause of progress in steelmaking processes and operations. A valuable contribution of this character is this comprehensive description of plant and practice at the steel-works of Australian Iron and Steel, Ltd., recently published by the Iron and Steel Institute, some features from which are given in this article.

IN the past ten years the ingot capacity of Australian steelworks has been increased greatly. To-day, 20 open-hearth furnaces have an annual capacity of 1,750,000 tons. Based on a 75% overall yield, this represents an annual consumption of 420 lb. of steel per head of population. Little information has appeared in steel literature abroad concerning Australian practice, some features of which have been developed to suit local conditions.

The Port Kembla plant of Australian Iron and Steel, Ltd., built since 1928, is the most modern in the country. The open-hearth shop of seven furnaces has an annual capacity of 750,000 tons of steel ingots. For economic reasons steelmaking at this plant has been based on a high-hot-metal furnace burden, which carries up to 450 lb. of iron ore per ton of ingots. The smelting of a rich ore, low in phosphorus and not low in manganese, in the blast furnaces provides a pig iron of excellent chemical and physical characteristics for basic open-hearth operations. In addition, the open hearth is supplied with limestone and lump manganiferous feed ore of high quality. These factors combine with flexible arrangement of suitable fuels fired to furnaces of modern design to ensure a practice notable for quality, high tonnage rates, and low cost. This plant is a unit of an organisation with ramifications wide enough to make it practically independent of the world's markets for all essentials to the manufacture of every type of high-grade steel.

Plant Layout

A cross-section of the open-hearth department, consisting of seven stationary furnaces, is shown in Fig. 1. In Table I is shown the nominal capacity of each furnace, the year of construction, the bath depth, and the flexibility of the plant with regard to fuel. It will be noted in Fig. 1 that the gas-house is 130 ft. from the melting shop building, and furnace gas mains pass beneath the intervening stockyard. Producer gas is made in eight 11-ft. diam. Wellman-Smith-Owen turbine-blown gas machines, each with an hourly gasification capacity of 7,000 lb. of good bituminous gas coal. The producers are paired to fire B, C, D and E furnaces, if so required. The coal required per week to operate four furnaces on mixed producer and coke-oven gas is 1,800 tons. A 5-ton gantry crane, operating with a suitable grab bucket, elevates coal from an adjacent 2,500-ton storage bin to the producer feed hoppers and loads out producer ashes as required. The gas-house is equipped with modern instruments and all the facilities necessary for the constant flow to the furnaces of gas of the required quality.

TABLE I.—OPEN-HEARTH FURNACE DATA.

Furnace.	Year Completed.	Nominal Capacity, Tons per heat.	Fuel.	Bath Depth, In.	Regenerators.	Tapping Launder.
A	1939	230	Tar and coke-oven gas only.	40	Double	Bifurcated.
B	1931	200	Producer gas and coke-oven gas.	38	"	"
C	1933	160		35	"	Single.
D	1935	160	Secondary fuel, tar and coke-oven gas.	35	"	"
E	1937	160		35	"	"
F	1940	230	Tar and coke-oven gas only.	40	Single	Bifurcated.
G	1941	280		44	"	"

The stockyard, 630 ft. × 100 ft., is served by two 15-ton combination magnet and crate-lifting cranes. A standard-gauge track runs through the yard, and the stocking cranes unload interworks scrap into pans of 36 cub. ft., arranged three to a crate. By lowering a motor-operated pair of lifting arms which engage the hooks on the crate, the cranes lift the stock to the charging benches opposite the furnaces.

The furnace bay is 765 ft. long, with a span of 80 ft., and the charging floor elevation is 18 ft. above ground level. The furnaces are charged by two overhead revolving type machines, each of 5 tons capacity. Hot metal is handled by a 100-ton crane with an auxiliary hoist of 15 tons.

At the western end of the building is a 1,200-ton Pennsylvania-type hot-metal mixer, which receives metal from 70-ton capacity transfer ladles filled from 120-ton Pugh-type ladles hauled from the blast-furnaces. The mixer is

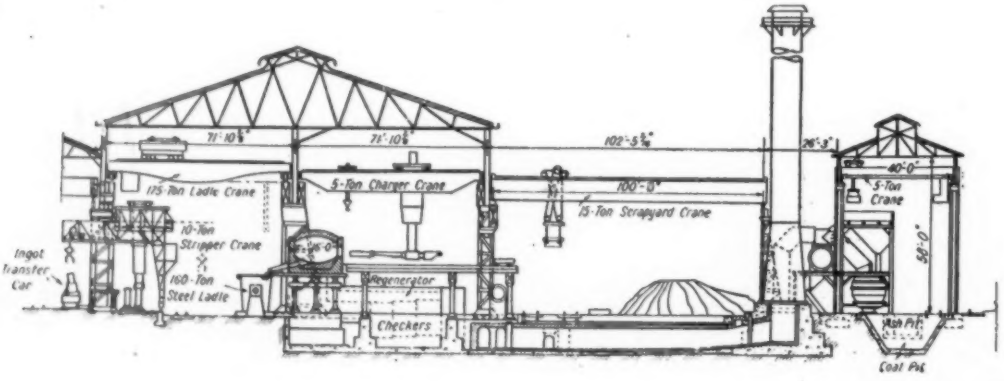


Fig. 1.—Cross-section through the open-hearth department.

ined with high-alumina bricks with a softening point of 1,680° C., which show no after-contraction when heated for 2 hours at 1,400° C. All iron delivered to the mixer and furnaces is weighed over a 100-ton bridge.

An electrically operated fettling machine runs on a broad-gauge track immediately in front of the furnaces. This machine is invaluable for " back-walling " units 18 ft. wide from bank to bank.

The main furnace columns are at 90-ft. centres, and experience on this plant indicates that the present trend abroad to build on 100—105-ft. centres for furnaces of 150 tons capacity or more is sound practice.

All furnaces comply with accepted modern standards relating to water-cooling, and are equipped with modified Blair ports and water-cooled straight-line gates and valves.

The casting bay, 945 ft. long and 67 ft. wide, is served by two ladle cranes of 275 tons and 175 tons capacity, respectively. There is also a 45-ton general-purpose crane. Tapped steel runs by either straight or bifurcated spouts into 160-ton capacity conical double-stoppered ladles of riveted construction. Sixteen ladles are in use. Ingots are cast into moulds on fixed stools in single line, and reference to Fig. 1 shows the arrangement of the two combination wall-type stripping and transfer cranes. These machines are supported on track-wheels on a monorail 3 ft. above pit-floor level, and on the soaking pits side by an overhead rail. Sufficient clearance above the stripping machine superstructure has been provided to allow freedom of ladle-crane movement.

Ingots are stripped and transferred between the openings of building columns direct to the soaking pits transfer car, which operates on a broad-gauge track parallel to the casting bay.

Fuel

Full advantage is taken of the fuel flexibility of the melting shop, as shown in Table I, to maintain tonnage rates, control fuel consumption, and improve repairs and maintenance costs. It is well known that an " aged " producer-gas-fired furnace responds in a remarkable fashion when switched to high-pressure fuels, viz., tar or oil. Records of this plant contain many instances of gas-fired furnaces, 26 to 30 weeks old, leaking badly through the midfeathers, being changed to tar and coke-oven gas and showing an immediate improvement in fuel rate of almost 50%. The tonnage per hour is also substantially increased, and this practice has an undoubted beneficial effect, not only on the open-hearth but also in the mills, because the flow of ingots is better balanced.

Hence, following the installation of a battery of Otto-Wilputte underjet-fired by-product coke ovens in 1938, the open-hearth department was completely piped for tar and coke-oven gas. The effect on steelmaking economy was immediate, and a programme of necessary modifications in furnace design was commenced at once.

Raw Materials

Compositions of the raw materials available to the open hearth are reproduced in Table II. Ore is drawn from the Whyalla deposits in South Australia and shipped approximately 1,070 miles to Port Kembla. Two modern unloaders transfer the ore into 60-ton steel bottom dumping cars, which are then hauled the 2 miles to the works. Most of the ore used in the steelmaking burden and feed is lump manganiferous, but the type of ore used is varied according to the grade of steel required. The ores are red hæmatites, low in phosphorus, and they range in manganese content from 0·5 to 30%. All open-hearth ore is screened to + 1 in. before use, the fines being incorporated in basic blast-furnace burdens.

The limestone is of exceptional purity and is free from adherent earthy matter. For open-hearth use it is well sized to 5—8 in. ring ; this does not agglomerate on the furnace bottoms as fines would do, and it comes up freely in the later course of the heat. Magnesite, from western New South Wales, has been used for many years by the Australian steel industry for bottom making. Preparations are being made to use this magnesite with chrome ore for the manufacture of metal-cased shapes for basic construction. Raw dolomite, mined at Mount Fairy, and crushed to ½-in. maximum, is used exclusively for slag-line fettling and making up door sills. Burnt dolomite, alone or mixed with magnesite, is used for rapid light repairs to bottoms and unusually impaired slag lines. During the past two years bauxite has replaced fluorspar as a flux in Kembla open-hearth operations. As a rule ore is used to break up lime, and the quantity of bauxite does not exceed 2 lb. per ton of ingots.

Furnace Burden

A typical burden sheet for a 230-ton heat of structural steel is of interest. The charge is as follows :—

Material.	Tons.	Total Metallics, %.
Mill return scrap	35	13·5
Mixer metal	195	75·6
Manganiferous ore (60% Fe)	46	10·7
Limestone	11	—
Re-carburisers	0·5	0·2

Limestone, 107 lb. per ton of ingots.

The outstanding features of such a furnace burden are the high hot-metal/scrap ratio, the heavy ore charge (448 lb. per ton), and the low flux rate. These furnaces, therefore, are subject to violent and protracted ore boils, resulting in a heavy carry-over of solid particles in the waste-gas stream. It is noteworthy that with such practice three of the furnaces have produced ingots at a rate exceeding 21 tons per hour.

Furnace Design

During the past ten years improvements in design have enabled large stationary furnaces to be built which carry such burdens continuously for campaigns exceeding 36 weeks. High production rates are maintained with low repair, maintenance and rebuilding costs. Compared with the first unit, which commenced to operate in 1931, the following features are incorporated in the construction of furnaces F and G, designed and built since 1939 :—

(*a*) Since the melting shop main building columns are on 90-ft. centres, bath areas could not be increased substantially without constricting critical areas necessary to generous end design. Hence, the average

TABLE II.—COMPOSITION OF RAW MATERIALS.
ANALYSES ON DRY WEIGHT.

Material :	Iron Ore.			Limestone.	Burnt Magnesite.	Burnt Dolomite.	Raw Dolomite.	Chrome Ore.		Bauxite.	Sea Sand.
Grade :	Foundry Basic Manganiferous.			5—8 in.	Crushed.	Lump.	Lump.	Lump.		Pebbles.	Windrift.
Origin :	Whyalla, South Australia.			Marulan, N.S.W.	Fifield, N.S.W.	Mount Knowles, N.S.W.	Mount Fairy, N.S.W.	Noumea, South Africa.		Mossvale District, N.S.W.	Port Kembla.
	%	%	%	%	%	%	%	%	%	%	%
SiO_2	3	3	3	1	6	2	3	5	2	3	96
Fe_2O_3	91	90	81	0·2	1	3	1	—	—	30	1
FeO	0·5	0	0	—	—	—	—	13	27	0·2	—
Mn	0·5	1·8	8	—	—	—	—	—	—	—	—
CaO	—	—	—	—	2	57	—	—	—	—	—
$CaCO_3$	—	—	—	98	—	—	55	—	—	—	—
Al_2O_3	3	3	3	0·3	1	3	0·3	20	24	44	2
MgO	—	—	—	—	89	34	—	13	1	—	—
$MgCO_3$	—	—	—	0·4	—	—	40	—	—	—	—
Cr_2O_3	—	—	—	—	—	—	—	46	45	—	—
TiO_2	—	—	—	—	—	—	—	—	—	5	—
P	0·05	0·05	0·05	—	—	—	—	—	—	—	—
H_2O	2	2	2	—	—	—	—	2	0	16	—

bath depths have been increased to enable furnaces to tap larger heats. Despite adverse criticism in some quarters abroad, the so-called deep-bath furnaces are an outstanding success in Australia and are noted for high tonnage rates and low costs.

(*b*) Modifications to furnace lines over the melting chamber, throat and ends have been effected to maintain a silica-roof life of 350 heats without retarding the steelmaking rate.

(*c*) A generous treatment of critical areas throughout the furnace and auxiliaries has been adopted to keep unit gas velocities within desired limits. This has had the desired effect of maintaining keener working furnaces and reducing costs of refractories.

(*d*) Basic ends were constructed to inhibit the fluxing of refractories and minimise slag-pocket accumulations.

(*e*) Suspended rigid roofs balanced for even distribution of stress were built.

(*f*) Two slag notches were provided to take care of flush slag.

(*g*) Slag pocket design was improved, and greatly increased capacities were introduced to enable furnaces to operate for long periods without stopping to remove accumulations.

(*h*) Regenerator design was developed to obtain satisfactory gas distribution and provide heating surface capacities commensurate with hourly tonnage rates. Stack temperatures were lowered and fuel figures improved.

(*i*) The two latest furnaces F and G (Figs. 4 and 5) were built with single slag pockets and regenerators. It is believed that the span of these arches approaches the largest yet tried. They are in excellent shape after two years' service.

(*j*) Modifications of checker-tile design and openings have been applied to facilitate routine cleaning and promote long-term maintenance of draught.

(*k*) Cleaning tunnels have been constructed beneath slag pockets giving access to the regenerator runner wicket, through which dust is either removed or blown towards the stack.

(*l*) Runners beneath regenerators have been made much larger.

(*m*) The flues between regenerators, valves and main stack are not filled in, and remain accessible. Flues and valves are placed in a well-illuminated chamber covered at valve-pit level with floor-plates. Numerous wickets provide quick access to runners and valve seats.

(*n*) The large furnaces have been equipped with bifurcated tapping spouts, splitting the heats into two ladles.

Results undoubtedly justify the efforts that Kembla has directed to the establishment of a balanced furnace.

Table III shows the development in design during the past ten years. It is evident that the producer-gas furnaces have not received as much attention as the fixed-fuel furnaces, but a campaign is now proceeding in their favour. Kembla B furnace was redesigned in July, 1941, to tap 200-ton heats, and it is the first gas-fired furnace in Australia to operate with fully basic ends—i.e., cold walls, end panels, and midfeathers of 100% basic construction down to the slag-pocket arches. At the time of writing, E furnace has commenced production with basic ends. The size of heat tapped remains the same, viz., 160 tons, hence sufficient data will be collected during the next twelve months to enable a sound forward policy to be formulated for the improvement of gas-fired furnaces.

To complete the general perspective of the salient features of Kembla furnace design, Tables IV and V show details of regenerators and slag pockets. Plant policy for regeneration is to provide up to 2,000 sq. ft. of heating surface per hour per ton of steel produced on fixed-fuel furnaces, and 3,000 sq. ft. on producer-gas furnaces. It is obvious that the gas furnaces at present are but moderately provided for, compared with contemporary furnaces abroad. Ample provision would involves total reconstruction, which conceivably might be undertaken in due course. The regenerator outlet temperatures of the tar furnaces are normal for units of this type, whereas the high figures of the gas furnaces indicate high-unit gas velocities and insufficient checker capacity.

TABLE III.—PROGRESS IN FURNACE DESIGN ABOVE CHARGING FLOOR. PERIOD 1931–41.

Furnace:	B.	B.	C.	A.	A.	F.	G.
Year built:	1931.	1931.	1933.	1939.	1939.	1940.	1941.
Figures taken:	1931.	1941.	1941.	As built.	1941.	1941.	1941.
Primary fuel:	Producer Gas.			Tar and Coke-oven Gas.			
Critical areas—							
Furnace laboratory centre-line	97 sq. ft.	118 sq. ft.	113 sq. ft.	127·5 sq. ft.	131 sq. ft.	127·5 sq. ft.	150 sq. ft.
At knuckles	50 "	54 "	55 "	48 "	71 "	82 "	72 "
Gas port area	6 "	4·2 "	4·2 "	—	—	—	—
Area over bridge wall	64 "	88 "	77 "	70 "	84 "	94 "	98 "
Total area:							
Air uptakes	38 "	68 "	53 "	69·5 "	77 "	94 "	80 "
Gas uptakes	12·7 "	14·4 "	12 "	—	—	—	—
Longitudinal dimensions—							
End steel to air uptakes	2 ft. 10 in.	3 ft. 3 in.	2 ft. 10 in.	3 ft. 3 in.	3 ft. 3 in.	3 ft. 3 in.	3 ft. 3 in.
Length—							
Air uptakes	5 ft. 6 in.	6 ft. 0 in.	6 ft. 0 in.	6 ft. 0 in.	7 ft. 0 in.	7 ft. 3 in.	6 ft. 0 in.
Throat	10 ft. 0 in.	5 ft. 0 in.	9 ft. 3 in.	7 ft. 0 in.	8 ft. 9 in.	7 ft. 0 in.	5 ft. 9 in.
Bath	43 ft. 0 in.	43 ft. 0 in.	43 ft. 0 in.	45 ft. 0 in.	45 ft. 0 in.	45 ft. 0 in.	47 ft. 6 in.
Overall furnace	79 ft. 8 in.	79 ft. 8 in.	79 ft. 8 in.	79 ft. 8 in.	79 ft. 8 in.	79 ft. 8 in.	79 ft. 8 in.
Elevations of roof from foreplate—							
(1) Centre-line heart to skew	4 ft. 4 in.	5 ft. 6 in.	5 ft. 3 in.	5 ft. 1 in.	5 ft. 8 in.	5 ft. 9 in.	5 ft. 9 in.
(2) Centre-line hearth to max. rise of roof.	6 ft. 7 in.	7 ft. 9 in.	7 ft. 9 in.	8 ft. 1 in.	8 ft. 8 in.	8 ft. 7 in. 11½ in.	9 ft. 2 in.
(3) At knuckles to skew	4 ft. 0 in.	4 ft. 6 in.	4 ft. 4 in.	3 ft. 7 in.	5 ft. 2 in.	4 ft. 9 in.	5 ft. 2 in.
(4) At knuckles to max. rise	5 ft. 3 in.	5 ft. 2 in.	5 ft. 10 in.	6 ft. 4 in.	7 ft. 2 in.	6 ft. 9 in.	6 ft. 8 in.
(5) Ends:							
(*a*) To skew	8 ft. 8 in.	8 ft. 10 in.	8 ft. 9 in.	8 ft. 6 in.	8 ft. 9 in.	8 ft. 9 in.	8 ft. 9 in.
(*b*) To max. rise	10 ft. 4 in.	10 ft. 10 in.	10 ft. 9 in.	10 ft. 6 in.	10 ft. 9 in.	10 ft. 9 in.	10 ft. 9 in.
(6) Elevation main roof minus that at knuckle	4 in.	1 ft. 6 in.	11 in.	9 in.	6 in.	10 in.	7 in.
(7) Elevation end of roof minus that at knuckle	4 ft. 8 in.	4 ft. 10 in.	4 ft. 9 in.	4 ft. 2 in.	3 ft. 7 in.	—	3 ft. 7 in.
Back-wall	Straight.	Straight.	Straight.	Sloping.	Sloping.	Sloping.	Sloping.
Monkeys, width—							
(*a*) Top	11 ft. 6 in.	8 ft. 10 in.	9 ft. 0 in.	10 ft. 0 in.	11 ft. 6 in.	11 ft. 3 in.	10 ft. 0 in.
(*b*) Bottom	11 ft. 6 in.	10 ft. 4 in.	11 ft. 0 in.	12 ft. 0 in.	13 ft. 6 in.	13 ft. 3 in.	12 ft. 6 in.
Length, roof—							
Hood	8 ft. 7 in.	11 ft. 6 in.	—	8 ft. 6 in.	9 ft. 0 in.	8 ft. 6 in.	8 ft. 7 in.
Hood slope	7 ft. 0 in.	5 ft. 0 in.	—	5 ft. 9 in.	6 ft. 0 in.	5 ft. 9 in.	4 ft. 9 in.
Knuckle	—	2 ft. 0 in.	—	2 ft. 6 in.	2 ft. 6 in.	2 ft. 6 in.	2 ft. 6 in.
Main roof slope	7 ft. 0 in.	5 ft. 9 in.	—	6 ft. 9 in.	5 ft. 6 in.	6 ft. 9 in.	7 ft. 9 in.
Main roof	29 ft. 0 in.	27 ft. 6 in.	—	29 ft. 0 in.	30 ft. 0 in.	29 ft. 0 in.	29 ft. 0 in.
Port nose to centre knuckle	4 ft. 6 in.	8 ft. 3 in.	7 ft. 5 in.	7 ft. 0 in.	8 ft. 9 in.	6 ft. 9 in.	6 ft. 3 in.

Table V emphasises the generous treatment of slag-pocket design in recent installations.

Furnace Construction

The general arrangement of the most recent Kembla open-hearth furnace is shown in Figs. 2, 3, 4 and 5. The complete superstructure of the furnace is rigidly bound together. Buckstays around the hearth proper are rolled 18 in. × 6 in. solid slabs, and 12 in. × 6 in. rolled steel joists are used on the ends.

The method adopted for rigid roof ties is illustrated in Fig. 2. In common with all furnaces in the plant, the skew-back channels are made up of specially rolled heavy-section semi-finished 24-in. × 7½-in. beams, with both flanges on one side oxy-cut away. The skew channels are swung from the roof superstructure by 1¼-in. diam. bolts. A 1-in.

water-cooling pipe is laid within the skew along the bottom flange, and extends from knuckle to knuckle along the front and back wall. These pipes are standard construction throughout the shop, and by the protection afforded to the skew channel in the later stages of a campaign they undoubtedly prolong the roof life.

The roof span is 20 ft. 1 in. over the furnace ends, and 23 ft. 1 in. over the sloping back wall at the centre-line of the furnace. The roof construction is alternate row-ribbed, and of silica brick, varying in thickness from 24 in. to 16 in. from the skew towards the crown, as follows :—

Eleven rows	24 in. and 20 in.
Four rows	20 in. and 18 in.
Balance of crown	18 in. and 16 in.

This type of roof averages about 350 heats, and has been found the best suited to rapid and effective hot repair. The elevation of the roof at the skewback channel above foreplate level is 5 ft. 9 in. at the tap-hole, and 5 ft. 2 in. at the knuckles. The sloping back wall is built of magnesite bricks encased in steel plate to within 18 in. of the skew. Chrome-magnesite pipes are then laid to the skew, and this work can be hot-repaired during charging operations if necessary.

Front walls are constructed entirely of chrome-magnesite pipes, with water-cooled frames and doors as standard equipment. Experiments with monolithic door linings are encouraging, and their extensive use in the future is indicated. Door arches are of 1-in. diam. chrome-magnesite pipes supported by 1¼-in. water-cooled coils laid behind the door frames.

Hearth Construction.—The nominal hearth dimensions are 47 ft. 6 in. × 17 ft., and the bottom is formed in a pan of 1-in. plate reinforced by arc-welded cover strips. The support is very strong and is made up of longitudinal 24-in. × 17½-in. beams on the concrete foundations, with cross beams of 12-in. × 8-in. section immediately beneath the pan. The hearth chills are fabricated and air-cooled.

In Australian steelworks there has been little departure from the orthodox in the installation of bottoms. There are probably two outstanding reasons for this. Excellent results have been obtained for many years with sintered magnesite bottoms. The oldest Kembla hearth is more than ten years old and is still giving good service. Again, Australian plants are far distant from the large steel producing countries, and are therefore cut off from the supplies of proprietary basic bottom-making materials so readily available to operators abroad.

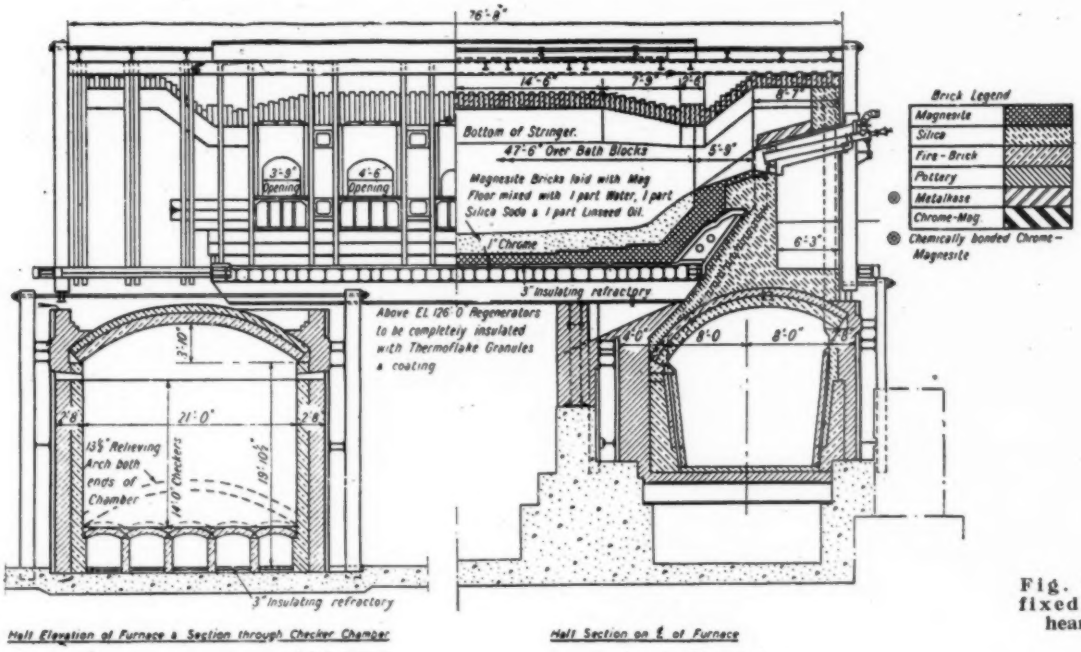

Brick Legend
Magnesite
Silica
Fire-Brick
Pottery
Metalkase
Chrome-Mag
Chemically bonded Chrome-Magnesite
Bottom of Stringer.
47'-6" Over Bath Blocks
Magnesite Bricks laid with Mag Floor mixed with 1 part Water, 1 part Silica Soda & 1 part Linseed Oil.
1" Chrome
3" Insulating refractory
Above EL 126'0 Regenerators to be completely insulated with Thermoflake Granules & coating
3'-9" Opening
4'-6" Opening
21'-0"
13½" Relieving Arch both ends of Chamber
14'0" Checkers
3" Insulating refractory
Half Elevation of Furnace & Section through Checker Chamber
Half Section on E. of Furnace

Fig. 2.—240-ton fixed-fuel open-hearth furnace.

TABLE IV.—REGENERATOR DETAILS.

Furnace:	B.		F.	G.
Date:	July, 1941.		July, 1941.	June, 1941.
Normal Fuel:	Producer Gas.		Tar and Coke-Over Gas.	Tar and Coke-Oven Gas.
Arbitrary Rating:	14 tons per hour.		17 tons per hour.	18 tons per hour.
Regenerator—	Air.	Gas.		
Length	28 ft. 0 in.	28 ft. 0 in.	28 ft. 0 in.	31 ft. 3 in.
Width	12 ft. 6 in.	8 ft. 3 in.	21 ft. 0 in.	21 ft. 0 in.
Height, checkers above rider tile	11 ft. 0 in. to 12 ft. 0 in. stepped.	11 ft. 0 in. to 12 ft. 0 in. stepped.	12 ft. 0 in.	13 ft. 6 in.
Checker work—				
Volume	3,880 cub. ft.	2,580 cub. ft.	7,060 cub. ft.	8,860 cub. ft.
Type	Box flue.	Box flue.	Box flue.	Box flue.
Tile	13 in. × 6 in. × 2 in.	13 in. × 6 in. × 2 in.	Beam tile 13½ in. × 6 in. × 3 in.	Beam tile 13½ in. × 6 in. × 4 in.
Hole, size	5½ in. × 5½ in.	5½ in. × 5½ in.	10½ in. × 4 in.	10½ in. × 6 in.
Total available heating surface	18,000 sq. ft.	12,000 sq. ft.	33,930 sq. ft.	36,760 sq. ft.
Size ratio air to gas	1·5 : 1·0		—	—
Combined volume	6,460 cub. ft.		7,060 cub. ft.	8,860 cub. ft.
Heating surface per ton per hour (one end only)	2,140 sq. ft.		1,995 sq. ft.	2,040 sq. ft.
Normal exit flue temperature	1,400°—1,500° F.		1,000°—1,500° F.	1,000°—1,050° F.

TABLE V.—DETAILS OF SLAG POCKETS.

Furnace:	B.		G.
Date:	1941.		1941.
Normal Fuel:	Producer Gas.		Tar and Coke-oven Gas.
Slag pockets—	Air.	Gas.	
Average length	27 ft. 4 in.	30 ft. 4 in.	26 ft. 6 in.
Average height	6 ft. 3 in.	6 ft. 3 in.	9 ft. 3 in.
Average width	5 ft. 0 in.	5 ft. 0 in.	13 ft. 6 in.
Total volume	1,900 cub. ft.		3,250 cub. ft.
Slag pocket deposit—			
Total per ton of ingots	13 lb.		15 lb.
Specific gravity	6·5		5·0
Weight per cub. ft.	406·25 lb.		312·5 lb.
Campaign tonnage before encroachment on free flow of gases	118,000 tons.		136,000 tons.

The installation of G furnace commenced by covering the pan rivets with crushed chrome ore. A 3-in. course of diatomaceous-earth insulating bricks were then laid. This was followed by 16½ in. of magnesite bricks, four courses on the flat and one on edge. These bricks were laid up with flour magnesite, linseed oil and water. The expansion allowed in the bottom and bank masonry was ⅛ in. every 18 in. laterally and longitudinally. Banks were constructed of magnesite brick over first-class fireclay brick, which began at a point 24 in. inwards from the steel of the hearth casing and then battered upwards. An area of 96 sq. ft. around the tapping hole was laid entirely in magnesite.

The total refractory thickness of this bottom is 36 in., made up as follows :—

	In.
Chrome ore on pan	½
Insulation	3
Magnesite brick	16½
Sintered magnesite	16

Furnace Ends.—The ends of G furnace received careful attention during design and construction. Reference to Table III will supply the outstanding features. The main objects of the design were, first, to balance the ends, if possible, to restrict the carry-over and keep the waste-gas velocity low, and secondly, to house and protect the tar

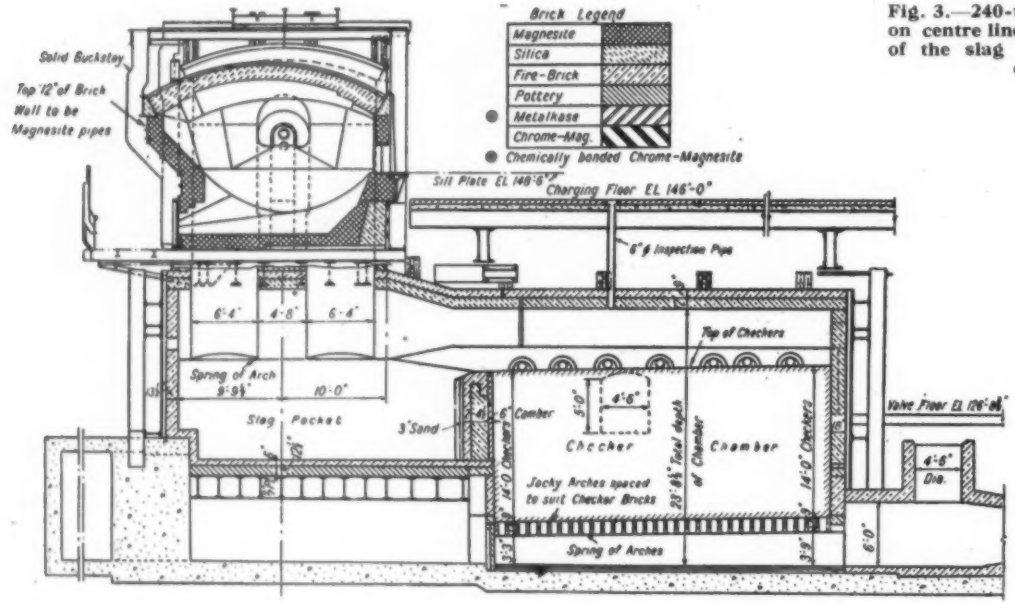


Fig. 3.—240-ton furnace. Sections on centre lines of the furnace and of the slag pocket and checker chamber.

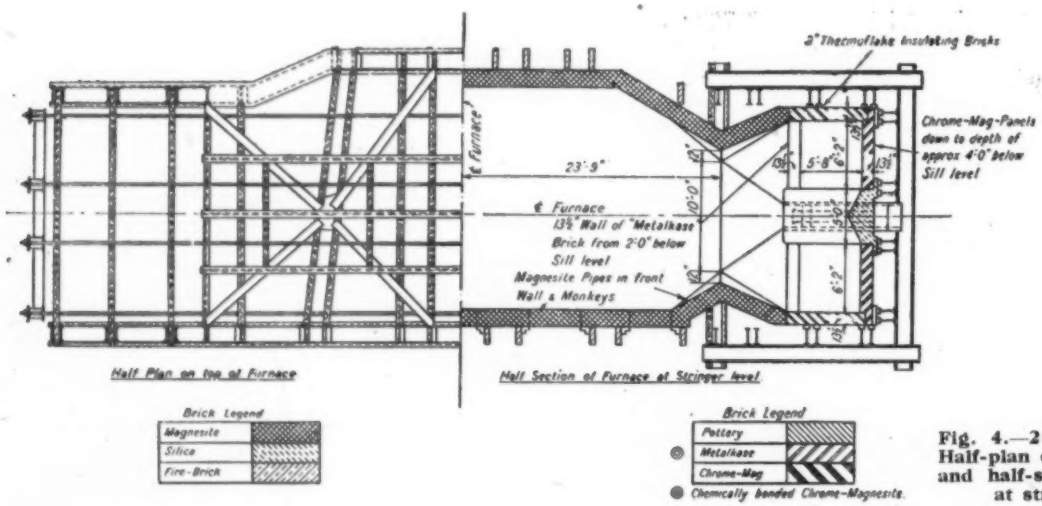


Fig. 4.—240-ton furnace. Half-plan on top of furnace, and half-section of furnace at stringer level.

The working bottom was sintered in with a mixture of six parts of crushed local magnesite and one part of basic slag. Continuous samples were taken at intervals to check the progress of this operation. Burning-in a bottom takes about eight days from firing the furnace to steel production, and 90 tons of magnesite are required. Prior to charging, the bottom is washed out with slag and "set back" for 8 hours.

The composition of G furnace bottom was:—

SiO_2.	FeO.	Fe_2O_3.	Al_2O_3.	CaO_2.	MgO.	P_2O_5.
%	%	%	%	%	%	%
7·4	1·5	5·0	2·4	9·0	73·5	0·2

Softening point, 1,850° C.

and coke-oven gas burner without encroaching on the desired free area between the bridge wall and roof.

A dog-box construction carries the modified water-cooled protection port. This port is 10 ft. long overall and 2 ft. 6 in. in width, and is simply an arched tank of soft-steel plate with a 3-in. layer of circulating water. Special pipes are welded into the nose of the port for routine flushing and the prevention of hot-spot. The port houses the combination tar and coke-oven gas burner, and in this instance the nozzle sizes are for tar ¾ in. diam., and gas 4¾ in. The burner is swivelled at the back so that the elevation and direction of the flame can be altered at will.

The end-panel and cold-wall construction is entirely basic down to the slag-pocket arches. Chrome-magnesite pipes 13½ in. long are used. An interesting feature of design is the excellent method used to support the great weight of the basic material and prevent the crushing of the silica beneath. At 3-ft. intervals T-bar cut from 20-in. rolled steel joist is welded horizontally to the buckstays to form trays upon which the refractory pipes are laid. Suitable expansion allowances are made at each level and no difficulties occur when heating up.

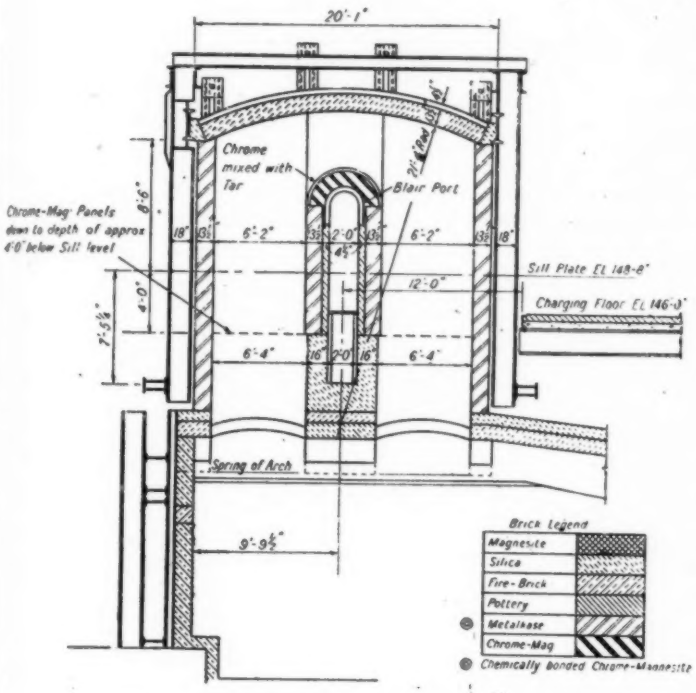


Fig. 5.—240-ton furnace. Section through the uptakes.

The downtake arches are of silica; this eliminates the old "jack" arches, affords greater free area for gas flow, and reduces repairs and maintenance.

Slag Pockets.—There are two features of pocket design worthy of mention, the first being the generous capacity allowed. Details are given in Table V. G furnace can operate for upwards of 580 heats, produced in a campaign exceeding one year, before pocket accumulation will interfere seriously with the free flow of furnace gases through the system. Slag-pocket floors are heavily sanded, and false silica walls without cavities are laid on the sides.

G furnace slag pockets are built on a 12½-in. reinforced concrete pad supported by 24 in. × 7½ in. rolled steel joists, which form the ceiling of a chamber 26 ft. 6 in. long × 16 ft. 9 in. wide × 6 ft. high. This chamber, entered by stairways at the ends of the furnace, gives access to the regenerator runners for blowing and cleaning. A 20-in. cool-air pipe laid longitudinally through the main furnace foundations from end to end keeps the atmosphere cool and comfortable to work in. Regenerator runners are therefore examined at intervals during operations, and cleaning proceeds as required.

Regenerators.—The 21-ft regenerator arches installed on G furnace have had almost two years' trial on furnace F. They are in excellent shape and condition. These arches are of fireclay brick. The fantail arches of silica have been streamlined into the regenerator chambers, with due regard for even waste-gas distribution over the checkers. Details of regenerator practice are shown in Table IV.

Valves and Stack.—The valves are straight-line and water-cooled with water-cooled seats angled in the flues to the stack. The flow of gases is reversed in 3 secs. The air valves are of cast-iron mushroom type, completely enclosed. Air for combustion is delivered by a fan of 23,000 cub. ft. per min. capacity at 3-in. water gauge, driven by a rheostat-controlled variable-speed motor.

The main stack flue, 8 ft. in diameter, is fully enclosed in a steel tube set in concrete and leads beneath the stock-yard to a firebrick-lined riveted steel stack 7 ft. diam. and 200 ft. high. The normal available operating draught at the base of the stack is 2-in. water gauge.

Insulation.—The insulation of open-hearth furnaces has not progressed very far in Australia, probably because published reports are rather arbitrary and conflicting, but the value of an air-tight furnace is well appreciated, and insulation is used extensively beneath charging-floor levels. G furnace regenerators and fantails are coated with 2 in. of mica-asbestos granules.

Water Cooling.—The furnace is cooled with sea-water delivered from a main which encircles the furnace root superstructure. Waste water is taken to the ends above roof level also. This system is well designed and completely safe.

Copper-Antimony-Nickel Gear Bronze

IN our December, 1942, issue attention was directed to a notable development in the production of a tin-free gear bronze by the Hamilton Gear and Machine Co., claimed to carry about 25% more load than tin bronze, and capable of running about 15° F. cooler at the same load. This material was stated to contain between 1·5 and 2·5% nickel and 7 to 8% antimony, the remainder being copper. An investigation on this material has since been made by the Phosphor Bronze Co., Ltd., and R. F. Hudson[1] has reported results obtained from which the following abstracts are taken.[2]

The metal proved to be fluid and easy running, and no trouble was observed in the production of sand and centrifugal castings. Most of the castings were poured at about 1,200° C. When the pouring temperature was lowered to 1,150° C., a skin appeared on the metal surface and the first traces of sluggishness were seen. The metal was clean, and top pouring was employed to ensure progressive solidification. In the case of chill-cast sticks, minute gas- or blow-holes were present in all the small diameters, suggesting that the alloy solidified so rapidly that air and mould gases were trapped. Castings in both green and dry sand had a good finish, and no metal penetration occurred.

"Best results were obtained by melting in oil-fired and coke-fired furnaces. No excessive oxidation or gas-absorption occurred. When natural-draft coke-fired furnaces were used, in conjunction with a slightly reducing atmosphere, 'spewing' occurred, the evolution of hydrogen forcing the molten metal to expand in the runner basin. Even when an oxidising flux (containing manganese dioxide) was used, there was still no tendency for the heads to sink. Many of the characteristics of the copper-antimony-nickel alloy were similar to those of 90-10 tin bronze."

The results of tests agree closely with the statements made by the makers, indicating that while the properties of the new alloy (with the exception of yield and hardness) are slightly lower than those of equivalent copper-tin-phosphorus alloys, they are sufficiently high to indicate useful commercial applications for the new material. The report also contains a discussion of the structure of the alloy, with typical photomicrographs.

The results so far obtained lead to the conclusion that "the alloy has favourable possibilities, especially with thin-sectioned sand-cast bushes and gear blanks and heavier sectioned castings made by the centrifugal process." The Phosphor Bronze Company is now carrying out service tests on both sand- and centrifugally-cast gearwheels, and is also studying the effect of heat-treatment of the alloy.

[1] R. F. Hudson. "Tin-Free Gear Bronze." *Foundry Trade Jour.*, 1943, vol. 69, pp. 115–118.
[2] *Nickel Bulletin*, 1943, vol. 16, No. 3.

A Review of the Work of the Ingot Moulds Sub-Committee

By R. H. Myers

As a result of experimental work and of statistical studies, it is clearly shown that great economy in ingot-mould material can be obtained very simply by reduced casting-to-stripping time, by attention to wall thickness and composition, and by improved layout equipment and practice in the casting pit. This review is presented here in a substantially abridged form.*

A CONSIDERABLE amount of work has been carried out by the Ingot Moulds Sub-Committee since its formation in 1934, and the review of this work, submitted by the chairman of this Sub-Committee, Mr. R. H. Myers, to the Committee on the Heterogeneity of Steel Ingots, includes, in addition to that already published, some unpublished work, as well as work in progress or contemplated.

The various aspects of the work are discussed in two main sections—one deals with the behaviour of ingot moulds in service, and the second with the basic causes of cracking. Each section is sub-divided. Thus, Section 1, on the behaviour of ingot moulds in service, deals with statistical investigation of performance and large-scale experiment.

At the outset a comprehensive list of some 47 factors believed to have an influence on mould life was prepared. The list included factors associated with both manufacture and use of moulds. The initial concern of the Sub-Committee was to make a selection of the more-important factors and to investigate the influence of each one separately.

It was realised that if any serious attempt was to be made to ensure comparable results, data would have to be expressed and experiments conducted on a common basis. To this end a standard form for recording mould history was prepared; this, together with a system in daily use by one firm, was published in the Seventh Ingot Report.

THE BEHAVIOUR OF INGOT MOULDS IN SERVICE

Altogether, data belonging to some 45 types of moulds were submitted and examined. The data appear in tabular form in the Seventh Ingot Report. Wide differences in mould life were noted between different types of moulds. This was not due to design as such (except perhaps in isolated cases), but to the treatment in service to which a particular design or type of mould was subjected. The small-end-up type showed an average of 83 lives as against 53 for the wide-end-up solid bottom and centre-run types.

Whilst the former type was used principally under car-casting and machine-stripping conditions, the latter suffered the disadvantages of pit—or floor casting., crane stripping, inversion, and in many cases bumping to release the ingot. They also took longer to strip, and would tend to be hotter when receiving the next cast. The average lives of open-end moulds teemed direct and bottom cast were 74 and 99, respectively. Tundishing improved the life of moulds, but this effect could not be isolated because of the deleterious influence of hot tops which were usually present.

The Effect of Casting-to-Stripping Time

Graphically, methods disclosed remarkable possibilities for increasing mould life with decrease in casting-to-stripping time. The rate of improvement in mould life was shown to increase rapidly the nearer one approached the minimum time practicable. Conversely, mould life was shown to decrease with increase of casting-to-stripping time up to a point where a further increase appeared to have no effect (Fig. 1). Both these aspects have been proved by practical experiment. Moulds occupying positions outside the band in Fig. 1 were shown to be abnormal, or subject to abnormal conditions.

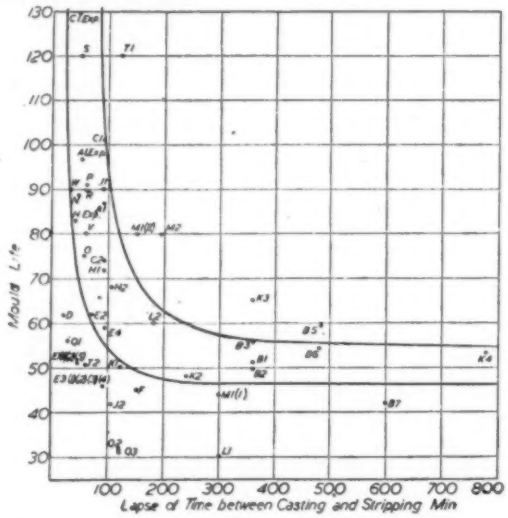


Fig. 1.—Correlation of mould life with lapse of time between the casting and stripping of ingots.

The true significance of decreasing the casting-to-stripping time does not seem to be appreciated. *The advantages to be gained justify large expenditure in equipping plants with stripping facilities where at present these are limited owing to plant layout.* With the exception of perhaps one or two specific types of steel, any depreciation in the quality of the ingot by quicker stripping has yet to be proved, once the frozen envelope is sufficiently strong to support the liquid centre, and always provided the ingot remains in a vertical position until solid throughout. At Hofors, in Sweden, equipment was installed to strip small high-quality steel ingots in 2–3 mins. after casting, with the result that the life of the ingot mould was trebled.

In further support of this statement, Jellinghaus, in discussing Ristow's paper on ingot moulds,[1] cited a case where the mould life had been increased from 205 to 310 by decreasing the average casting-to-stripping time from 18 to 8 mins.

Earlier release from hot ingot reduces the temperature attained by the mould, and in turn the tendency to oxidation, growth and crazing of the working face, as well as to premature crack failure; it also renders the mould cooler for subsequent use, thereby making it possible to reduce mould stocks; and, not least important, it permits the delivery of hotter ingots to the soakers.

An example of what can be spent with advantage on plant and equipment to ensure better mould performance is given below. Consider a steelworks producing 250,000 tons of ingots per annum, the cost of ingot moulds being £14

* Paper No. 19/1943 of the Committee on the Heterogeneity of Steel Ingots. Published by the Iron and Steel Institute, March, 1943. (Advance copy.)

1 *Stahl und Eisen*, 1940, vol. 60, pp. 401–404, 427–433.

per ton, less a scrap value of £6 per ton (i.e., £8 per ton net). Assume that the ratio of mould to ingot weight is 1·0, and that an average mould life of 100 casts can be increased to 130 casts. In the author's belief, a 30% increase is a conservative estimate of what could be achieved in many high-production plants :—

Mould life	100 casts	130 casts
Consumption of mould metal per ton of steel	22·4 lb.	17·2 lb.
Cost of moulds per annum	£20,000	£15,385
Cost of mould metal per ton of steel	19.20d.	14.77d.

By increasing the average mould life from 100 to 130 casts there is therefore a saving of 4·43d. per ton of steel, or £4,615 per annum. At an interest rate of 5% this would justify the expenditure of about £92,000 on plant and equipment to secure this extra mould life, to say nothing of the added advantages obtained, such as hotter ingots sent to the soakers, less mould stocks required, etc.

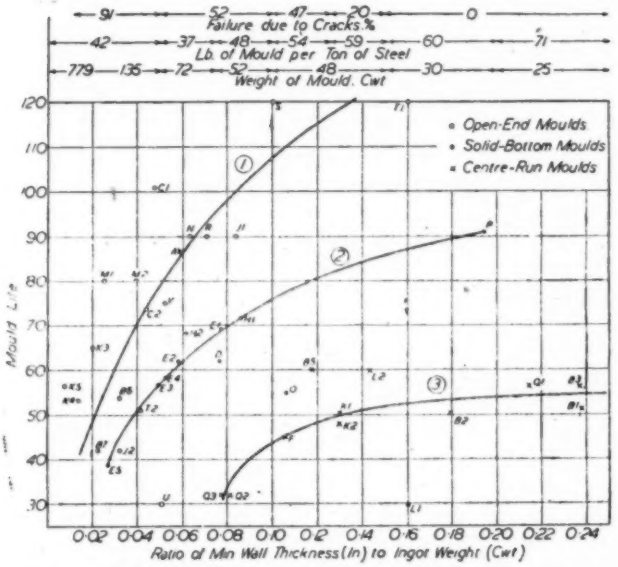

Failure due to Cracks %
Lb. of Mould per Ton of Steel
Weight of Mould. Cwt
Open-End Moulds
Solid-Bottom Moulds
Centre-Run Moulds
Mould Life
Ratio of Min Wall Thickness (In) to Ingot Weight (Cwt)

Fig. 2.—Correlation of mould life with the ratio of the minimum wall thickness to the ingot weight.

The Influence of Mould-Wall Thickness

A graph of exceptional significance is shown in Fig. 2. The factor selected for plotting against mould life was the ratio of the minimum wall thickness to the ingot weight. The mould ratio (cross-sectioned area of the mould to that of the ingot) when plotted against mould life showed similar, but slightly less definite, relationships. This graph clearly indicates not only the relationship between the wall thickness and the mould life for three different types of moulds, but also the dependence both of the mode of failure and of the consumption of mould material on the wall thickness as follows :—

(i) Thick-walled moulds of high mould ratio give a high life and fail by crazing.

(ii) Thin-walled moulds of low mould ratio give a low life and fail by cracking.

(iii) There is a strong tendency (in the examples given) for the large moulds to crack through having a low mould ratio, and for the small moulds to craze through having a high mould ratio.

(iv) At some intermediate mould ratio, where the mode of failure is presumably on the border line between crazing and cracking, minimum consumption of mould material is obtained.

The optimum wall thickness for a given mould capacity will vary with the design and the composition of the mould, and particularly with the conditions under which the mould is used.

In general, if a mould is failing consistently by crazing, the wall thickness is above the optimum, and steps can be taken to reduce it, with resulting economy in mould material. Conversely, if a mould is failing consistently by cracking, the wall thickness is below the optimum, and thickening the mould wall will again effect economy in mould material.

The Effect of Composition on Mould Life

A recent examination by graphical methods of some four hundred moulds enabled the United Steel Companies, Ltd., to decide upon the optimum composition (manganese 0·8–1·0% ; silicon, 1·0% ; sulphur, 0·06% max. ; phosphorus, 0·10%) for one type of mould in frequent use (Figs. 3, 4, 5 and 6). The moulds were arranged in groups according to increasing contents of the above elements (the number in each group is indicated on the life curve), and by plotting for each group the average value of mould life, the effect of these elements on the life was observed. The results were reported at the Seventeenth Open-Hearth Conference, and are embodied in the following sub-sections (*a*) to (*d*) :—

(*a*) *The Effect of Manganese.*—In the early stages of the work it was shown that as the manganese increased from 0·75 to 1·00%, failure of the mould, due to cracking, decreased and the life increased.[2] Unlike silicon, which is a graphitiser, manganese is a carbide stabiliser, and on the above result it would not have been unreasonable to expect higher manganese to induce rather than to reduce cracking. Having in mind, however, the relatively high range of silicon (1·8 to 2·0%), it can be assumed that any influence of the manganese as a stabiliser has been masked by the high silicon as a graphitiser, and the tendency to reduce cracking is probably the result of additional strength imparted to the material by the manganese. In a later examination of records, where a lower silicon content was found to improve life, the optimum manganese was shown to be lower, of the order of 0·9% (Fig. 3). Naturally, the stabilising effect of manganese, with its tendency to reduce crazing, was more marked with the lower silicon, and a lower concentration of manganese was sufficient to produce the same effect.

(*b*) *The Effect of Silicon.*—Within the range of 1·70 to 2·00% of silicon, the mould life was shown to decrease with increase in silicon—i.e., oxidation, growth and crazing were liable to occur at an earlier stage. Conversely, the life increased with decreasing silicon content, despite a greater tendency to crack—rather than to craze-failure.[3]

Later, a similar effect on the life was observed over a wider range of silicon, from about 0·9 to 1·9% (Fig. 7). Below about 0·9% of silicon, however, the life commenced to decrease again, whilst at 0·6% of silicon premature failure by cracking was observed (Fig. 4). Although more moulds failed by crazing the higher the silicon content was, some moulds failed solely by cracking and some solely by crazing at all values of silicon between 0·9 and 1·9%, but for a given silicon content those which failed by crazing gave a higher life than those which failed by cracking (Fig. 8). Those moulds, however, which showed a tendency to fail by cracking and crazing simultaneously gave the highest life throughout. This is not shown on the graph, but was found on a subsequent examination.

(*c*) *The Effect of Sulphur.*—The adverse effect of sulphur above 0·045% (Fig. 5) was apparent but this was not necessarily due to sulphur as such, but to poor coke and cold-working of the cupola.

(*d*) *The Effect of Phosphorus.*—The increasing beneficial effect of phosphorus up to 0·1% (the limit of the range examined at the time) was pronounced (Fig. 6). Two moulds since made, having phosphorus contents of 0·216% and 0·245%, well exceeded the average figure for mould life. Moulds made with more than 0·3% of phosphorus, however, gave lives less than the average, and failed prematurely through cracking. Moulds containing 0·17% and 0·24% of phosphorus (mentioned in the Sub-Committee's First

2 Seventh Ingot Report, pp. 153–154.
3 Seventh Ingot Report, p. 154.

Report[4]) gave much improved lives when compared with those with 0·075% of phosphorus.

It would, therefore, seem that for the individual condition prevailing at this plant it was safe and even beneficial to increase the phosphorus content of mould iron to 0·25% with the possible exception of large moulds for forge

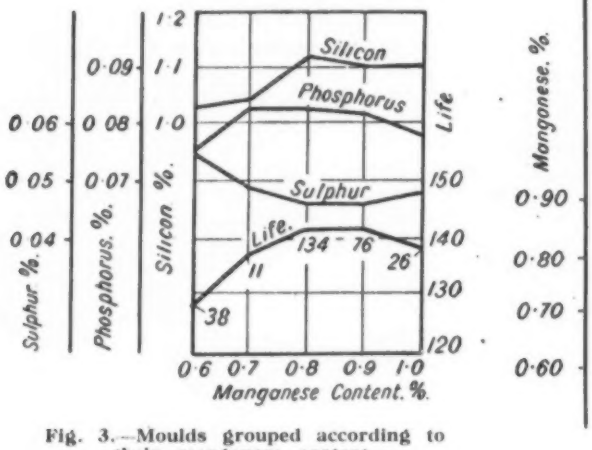


Fig. 3.—Moulds grouped according to their manganese content.

products. It is especially emphasised that high-phosphorus mould scrap should be segregated and clearly marked to prevent it from being remelted for manufacture into large ingot moulds.

(*e*) *Effect of Chromium.*—Additions of 0·3% and 0·5% of chromium were made to cupola charges of 100% hæmatite iron. Compared with moulds made from 100% hæmatite iron alone there was some slight improvement in life, but this was more than offset by the increased cost of the mixture.

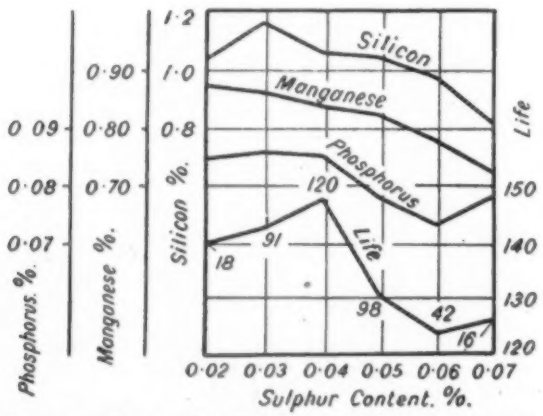


Fig. 5.—Moulds grouped according to their sulphur contents.

The Influence of Foundry Practice

The total make of one type of mould for the year was arranged in groups according to the month of manufacture and the average analyses. There was a close relationship between the phosphorus content and the life except at one period, when the sulphur increased. Subsequent investigation proved that this was not due to sulphur as such, but to indifferent cupola practice, the bad period being associated with poor quality coke and cold melts. A stricter control of foundry practice was, therefore, imposed, and a system of recording put into effect. Attention is directed to the method of recording in the foundry; a fluidity test for judging the suitability of cupola iron for ingot moulds; the relationship between the latter fluidity test and the notched spiral fluidity test; the effect of various proportions of mould and steel scrap in the cupola charge; the effect on compositions of repeatedly melting mould scrap in the cupola; the effect of different proportions of various brands of hæmatite, basic iron, mould, foundry and steel

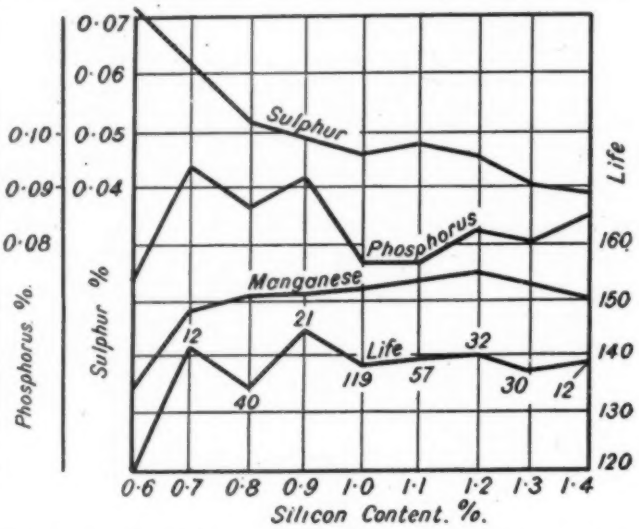


Fig. 4.—Moulds grouped according to their silicon contents

scrap; the effect of part-blown hæmatite iron from the converter; the effect of the time between casting the mould in the foundry and its removal from the sand; the effect of annealing; the effect of weathering to reduce foundry stresses.

Modes of Failure of Ingot Moulds

Special reference is made to crazing and premature cracking of ingot moulds. In the initial stages, Mr. J. G. Pearce[5] explained in detail the mechanism by which

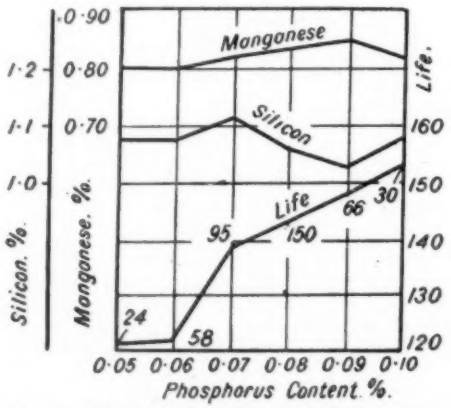


Fig. 6.—Moulds grouped according to their phosphorus contents.

crazing commenced, and proceded to the ultimate failure. This was due to oxidation and growth commencing in the graphite cavities, and was accelerated when the working face reached the pearlite change-point at a temperature of about 780° C., when a definite contraction took place. Thus, contraction of the working face would take place simultaneously with expansion immediately behind, giving rise at first to the formation of minute fissures and develop later the well-known crazy appearance.

Mould irons for withstanding cracking should contain coarse graphite and be of relatively open grain; mould irons to withstand crazing should be harder and of a finer graphite structure. The two requirements are antagonistic,

4 Seventh Ingot Report, Section VI.
5 Seventh Ingot Report, pp. 184–195.

with the result that mould iron at its best is only a compromise. Reference was also made to the graphitising as well as to the carbide-stabilising constituents of the mould iron, such as silicon, aluminium, nickel and copper on the one hand and chromium, molybdenum and manganese on the other.

Premature Cracking.—Cracks, whether of the "abrupt" or gradually developing type, are obviously due to the uneven distribution of temperature, causing stresses in excess of the strength of the material at some point and time during the heating cycle. Points of weakness, at which cracks develop, may be due to bad design (abrupt changes in section); to residual stresses, the accumulation of kish, porosity or minute cracks, due to indifferent founding; or to heterogeneity.

Liability to cracking may be somewhat minimised by employing a relatively soft iron (high carbon, coarse graphite) which can accommodate itself more readily to stress than one of harder quality. For this reason a softer iron is used for moulds of large capacity where failure due to cracking must be avoided. Investigatory work in this field on moulds which have failed in service by major cracking is briefly described.

THE BASIC CAUSES OF CRACKING

Whilst the causes of cracking can be expressed qualitatively, there are still insufficient quantitative data available on which to design moulds on a scientific basis with the object of eliminating cracking. Until data are available, moulds will continue to be designed by empirical methods.

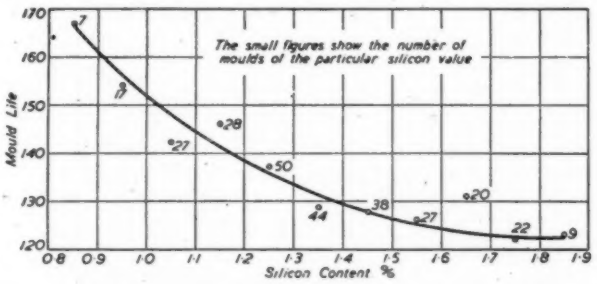


Fig. 7.—22½-in. semi-closed-top moulds. Effects of silicon content on the life.

The Sub-Committee soon realised, therefore, that fundamental work in this field should proceed concurrently with the more-practical aspect, and as a first step instituted the Stresses in Moulds Panel to direct this side of the work, which is summarised below.

The Determination of the Temperature Gradients in the Wall of a Slab Mould

Only a very imperfect picture of the temperature distribution in a mould can be obtained from the known physical constants, and, since other important factors play a part, the Sub-Committee decided as a first step to obtain these data experimentally. The investigation was carried out on a 10-ton slab mould. Time-temperature curves were obtained from no fewer than 32 positions in adjacent walls, which included the corner. The technique employed and the results obtained are given elsewhere.[6] The results show *inter alia* that:—

(i) Quite apart from the end effect, a wide variation in temperature exists between different parts of the working face of the mould. This is accounted for under (iii) and (iv), as well as by the greater mass effect of mould material at the corners.

(ii) There is a tendency for the broad side of the mould to bend inwards, bringing pressure to bear on the broad side of the ingot, particularly in that region where crazing is at its worst.

(iii) There is less overall lateral contraction of the ingot and less lateral expansion of the mould betweene the two broad sides compared with the other two.

(iv) There is a considerable early withdrawal of the ingot from the mould at the corners and narrow sides, and no indication of this in the middle of the broad side.

The results as a whole suggest that before the stresses in a mould can be accurately calculated for any mould the temperature distribution must be determined experimentally. Data were also required on the physical and mechanical properties of mould material at room and at elevated temperatures.

The Thermal Relations between Ingot and Mould

Mr. T. F. Russell, who had already done considerable mathematical work in this field, was induced by the panel to publish his earlier work together with that undertaken later on behalf of the Panel.[7]

His paper includes a quantitative examination of Saitô's formulæ for the distribution of heat between ingot and mould.

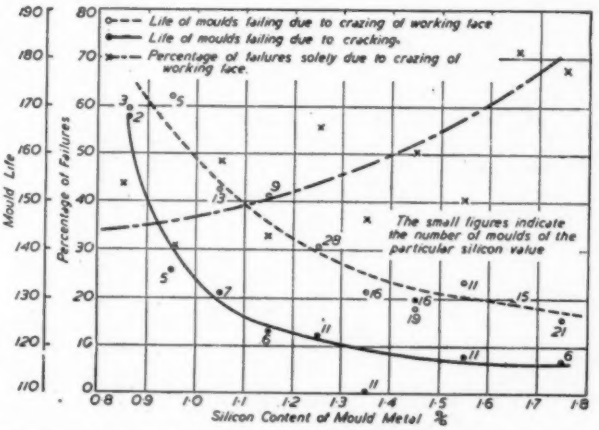

Life of moulds failing due to crazing of working face
Life of moulds failing due to cracking.
Percentage of failures solely due to crazing of working face.
The small figures indicate the number of moulds of the particular silicon value
Mould Life
Percentage of Failures
Silicon Content of Mould Metal %

Fig. 8.—22½-ton semi-closed-top moulds. Effect of silicon content on the mode of failure.

Examples are worked out for circular and square ingots of the same cross-sectional area cast into moulds of different thicknesses. Curves show the temperature at different points in the ingot and moulds, the temperature distribution across a diameter and the total quantities of heat in the ingot and the mould at different times.

Curves are also drawn to show the effect of the mould ratio on the temperature cycle occurring in the ingot near to the mould wall, and on the time taken for the temperature at the centre of an ingot to fall by certain amounts representing solidification. The latter indicate that solidification is accelerated by increasing the mould thickness until the mould ratio is about 0·8—1·0, but that a further increase has an inappreciable effect. Four sets of experimental results on the measurement of mould temperature are examined, and they show that the greatest difference between theory and practice is found at positions in the mould near the working face, due doubtless to the effect of the air-gap forming between the ingot and the mould.

Thermal Stresses in Ingot Moulds.—Work done by Mr. T. Land follows in logical sequence upon that of Mr. Russell. His paper[8] includes the calculation of the stresses in a long circular mould, on the assumption that the physical

6 Eighth Report on the Heterogeneity of Steel Ingots, pp. 278–294.

7 *Jour. of Iron and Steel Inst.*, 1941, No. I, pp. 163P—191P.

8 Paper No. 20/1943 of the Committee on the Heterogeneity of Steel Ingots. Iron and Steel Institute.

properties of the mould iron are independent of temperature and of stress. Under these conditions it is shown that stresses at the outer surface are roughly equal to the ultimate strength of the mould iron. In a very thin mould however, the stresses exceed those in one of normal thickness by about 20%.

Next, the theory is developed to take account of the plastic deformation and varying physical constants of the mould iron, and in these circumstances the stresses are shown to be approximately half the ultimate strength of the mould material. Other sections deal with moulds of non-circular section and with end effects. Whilst a further section points out that the surface stresses can be calculated from the measurement of the temperature and expansion of the outer mould surface, and the exact relationships are set out together with suggestions for further experimental work before expansion of the present work can be attempted. The last section gives the detailed mathematical analysis of stresses in long circular moulds of perfectly elastic material.

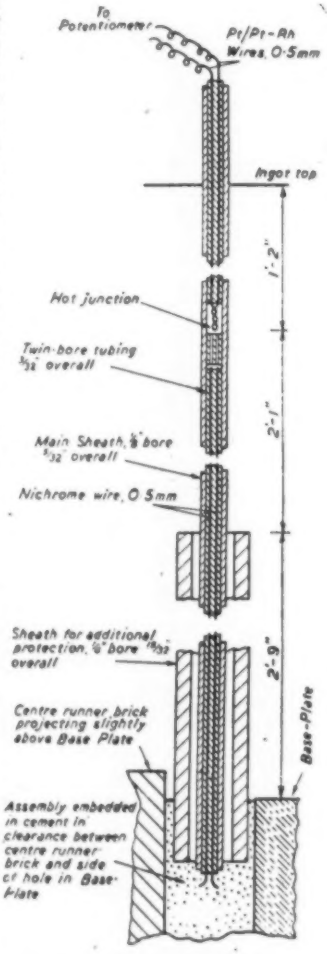

To Potentiometer
Pt/Pt-Rh Wires, 0·5mm
Ingot top
1'-2"
Hot junction
Twin-bore tubing 5/32" overall
2'-1"
Main Sheath, ⅛" bore 5/32" overall
Nichrome wire, 0·5mm
Sheath for additional protection, ⅛" bore 5/32" overall
2'-9"
Centre runner brick projecting slightly above Base Plate
Base-Plate
Assembly embedded in cement in clearance between centre runner brick and side of hole in Base-Plate

Fig. 9.—Assembly of thermocouple for ingot-temperature measurement.

Laboratory Work

Reference is made to work done on the mechanical and physical properties of mould material; to work done on model moulds; and to further work contemplated. The latter is briefly indicated by the following subheadings: representative material; stress-strain curves; Poisson's ratio; and large-scale experiment involving the measurement of strain and temperature in the wall of a commercial mould into which an ingot has been cast.

A Technique for Determining the Temperature in the Interior of a Freezing Ingot

In the large-scale experiment envisaged for the determination of heat distribution, it was considered desirable to know accurately the temperature conditions within the freezing ingot. A technique was devised and a works experiment was conducted by the author of this review through the courtesy of the late Mr. R. P. Smith. Fig. 9 is a diagram of the assembly. The couple encased in a slender silica sheath was anchored to the base-plate, and was thus supported in the ingot mould in a vertical position owing to buoyancy. The experiment was conducted on a 48-cwt. uphill-cast ingot, using a silica sheath of only $\frac{5}{32}$-in. outside diameter. Thus, the time lag was reduced to a minimum. A time-temperature curve was obtained for that position in the freezing ingot where it is most difficult to maintain the silica sheath intact, viz., in the last portion to freeze immediately below the head.

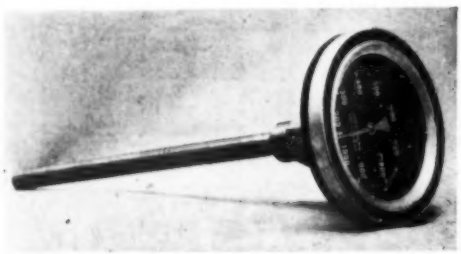
Heavy Industrial Dial Thermometers

FOR general industrial use the glass and mercury thermometer has the disadvantage of being easily broken, and when exposed continuously to relatively high temperature conditions the mercury thread tends to stick and thus inaccurate readings are likely to result. Under present conditions replacements are difficult, and breakages are a serious matter.

To reduce these difficulties special dial thermometers are available, and much interest is attached to the three types made by the British Rototherm Co., Ltd., comprising the co-axial, vertical, and flange types. These have a metal stem and a metal dial with heavy glass front and chromium-plated bezel, like the small clock or pressure gauge, with the pointer moving over a graduated temperature scale, which can be read easily from a considerable distance. Each of these types has three standard sizes of dial, $2\frac{1}{2}$ in., 4 in., and 7 in. diameter, with two standard stem diameters, $\frac{5}{8}$ in. and $\frac{7}{16}$ in., the stem also being made a length to suit the application.

The co-axial design has the dial at right-angles to the stem, as in the accompanying illustration, while in the vertical type the dial is parallel with the stem; the flange type with a co-axial dial is particularly suitable for ovens, a gland being provided for double-walled ovens, tanks or other equipment, the gland making a secondary joint with the inner wall, and a main joint with the outer wall.

The temperature ranges are divided into two groups—that is, a series of graduations from 20° F. to 500° F.—with ten different dials available, and a high temperature series over 500° F., in five groups up to 1,000° F.



Industrial dial thermometer of the co-axial type, graduated for a temperature range of 200°–1,000° F.

It may be mentioned that the co-axial model has the advantage that the small bi-metal multi-helical coil, the expansion or contraction of which is proportional to the temperature, is coupled direct to the dial pointer without the necessity of using a gear-box, so as to convert the movement of the light connecting shaft in the stem from the coil at the bottom into the horizontal movement necessary to move the dial pointer at the top.

These instruments are suitable for almost any gas or liquid up to a temperature as high as 1,050° F., so that in many cases they are convenient for boiler flues, both before and after the economiser. They are also suitable for core ovens and for ovens designed for the treatment of light alloys. So far as immersion in liquids is concerned, the only minor trouble is that no metal is entirely resistant to every known liquid, but for most cases the standard material is suitable, although stainless steel or other highly resistant material can be used if required.

The Wellman Smith Owen Engineering Corporation, Ltd., announce that Sir Walter Benton Jones, Bt., Mr. Allan C. Macdiarmid, Mr. John E. Jones, and Mr. G. S. McLay have joined the Board of Directors of that company.

Calcium and Some of its Applications

By L. Sanderson

One of the alkaline "earths'" calcium has many uses. Apart from being the basic element in an important group of compounds, it has many applications in the metallic state either in a relatively pure form or as an alloy, and some interesting features associated with this element are discussed.

CALCIUM is a soft white metal with an atomic weight of 40·07, specific gravity 1·58, melting point 810° C., boiling point about 1,400° C. (some authorities give 1,240° C. Its atomic number is 20; tensile strength approximately 512 kg./sq. cm.; heat of combustion per gram. equivalent, 80·097 cal.; specific heat between — 180 and + 20° C., 0·1574. Reacting vigorously when heated in the presence of oxygen to form the oxide, it also reacts with water to form the hydroxide. It is difficult to file or saw, because it clogs the teeth of these tools, but it is readily machinable, can be cold-hammered into thin sheets, and swaged into rods. It rapidly tarnishes to a yellow hue when exposed to the atmosphere.

Calcium does not occur free in nature; it is, however one of the most abundant of elements, and is estimated to comprise 3·47% of the earth's crust. It is an essential constituent of a large number of rock-forming minerals. The carbonate is widely distributed as calcite, chalk, limestone, marble, coral, etc. It is also found in conjunction with magnesium carbonate in the form of dolomite, or, when crystallised, as bitter-spar. There are, however, numerous other forms—e.g., scheelite (one of the calcium tungstates—$CaWO_3$); gypsum ($CaSO_4.2H_2O$), one of the phosphatic minerals; and many arsenates, antimoniates, niobates, tantalates, vanadates, borates, and silicates.

Salts of calcium are discovered in the waters of natural springs, generally calcium sulphate or calcium hydrocarbonate, and similar salts, as well as calcium phosphate and calcium fluoride are also found in sea-water. Calcium is, of course, an essential in human food, and the quantity necessary for ordinary normal consumption averages, it is estimated, about 0·45 grm. per 70 kgms. of body weight per day.

Numerous efforts have been made to extract calcium metal chemically, but these have broadly failed, though a virtually pure metallic calcium is obtainable as a result of reducing calcium iodide with metallic sodium. Electrolysis, in which calcium chloride solution, used with a mercury cathode, has produced an amalgam difficult to separate, and the calcium metal when obtained is in powder form. Another electrolytic process comprises a fused mixture of sodium and calcium chlorides, also with calcium iodide at a low red heat, employing a cathode of nickel and an anode of graphite. The basic commercial method is, however, the electrolysis of calcium chloride or a mixture with fluorspar, using a graphite anode in a refractory vessel.

Because they are extremely active chemically, calcium alloys have played and continue to play an important part in the manufacture of metals. Calcium readily combines with oxygen and other gases, and at elevated temperatures reduces virtually all metallic oxides. As a result, it can be considered a most efficient deoxidiser, degasifier, and reducing agent for metallurgical operations. It is, as a rule, alloyed with other substances—e.g., silicon, manganese and aluminium, because this is more convenient, and because the combinations offer advantages not possessed by straight calcium. Three typical combinations are calcium-silicon, calcium-manganese-silicon, and calcium-aluminium-silicon, whose function it is desirable to outline.

Calcium-silicon is employed in the ladle to deoxidise and degasify both steel and cast iron. Both elements of this alloy are active deoxidisers and form a slag with a low fusion point that quickly rids itself of molten metal. Used as a ladle deoxidiser for cast steel, calcium silicon cuts out deleterious chain-type sulphide inclusions, and in consequence elevates the yield-point and raises the reduction of area of the steel. The fluidity of the molten steel is considerably improved by the employment of this alloy, when added in amounts of 2 lb. or more per ton. It is employed in basic electric furnace practice to develop with maximum rapidity a highly fluid reducing slag. It is likewise employed in manufacturing high-quality alloy steel, stainless steel, and cast iron of specially high strength. Calcium-silicon contains between 28 and 35% of calcium, according to the grade, and 60–65% of silicon.

Calcium-aluminium-silicon combines three of the most active elements employed as deoxidisers and degasifiers of steel. The deoxidation products thereby formed have a comparatively low fusion point, and quickly coalesce into large particles that rise swiftly in the molten steel. As a result a well-killed steel exceptionally free from non-metallic inclusions is produced. This alloy contains 10–14% calcium, 8–12% aluminium, and 50–53% silicon.

Calcium-manganese-silicon combines three active elements, whose combination forms an admirable scavenger for oxides, gases, and non-metallic impurities in steel. Its use improves the transverse strength in forged and rolled steel. Excellent ductility has been obtained with steel castings treated with this alloy. It contains 16 to 20% calcium, 14 to 18% manganese, and 55 to 60% silicon.

Calcium alloys have also been put forward as bearing metals. The use of the compounds is so well known as hardly to need mention. Lime or limestone is employed in making mortars and cements, sand-lime bricks, glass and glazes, in purifying water, soda manufacture, bleaching powder, acetylene gas, the purification of coal gas, the making of insect killers and sprays, sugar manufacture, wood distillation, paper-making, and tanning.

Calcium is introduced into certain light aluminium alloys and is also a hardening component in a number of lead-base anti-friction alloys. It can be employed as a reducing agent to eliminate the oxide of numerous rare metals—e.g., uranium, zirconium, titanium, etc. It is also employed in the hot-cathode rectifier to clean up the argon gas. The tungstate is employed in varying degrees of fineness for the preparation of screens for X-ray photogrgphs. The sulphide (CaS Aq.) is employed in the preparation of luminous objects—e.g., match-boxes, watch dials, and house number-plates, and also for preparing As-free H_2S. The hydride formed by heating the oxide in a current of hydrogen to 700° C. reacts with water, liberating hydrogen, and has been employed as a portable source of hydrogen for balloons. The bisulphate is widely used in making sulphite cellulose and paper.

In the electrolytic production of metallic calcium, a graphite shell contains the fused electrolyte, which is cooled on the bottom by a water-jacket. This maintains a solid mass of electrolyte on the bottom. A jacket of fine graphite and lamp black protects the shell. Two water-cooled graphite plates form the anodes. On electrolysis a layer of metallic calcium is formed at the lower end of the rod on the surface of the electrolyte. The rod is gradually raised, the thickness of the layer increases, and eventually a rod of metallic calcium is formed as a continuation of the iron cathode. Thickness of this rod is maintained at about 2 mm. by regulating the speed of the motor to the resistance of the calcium electrode. Rods 35 cm. long and 7·5 mm.

diameter are obtained. If rolling or drawing down these rods produces brittleness as a result of excessive manipulation, a brief annealing at 350° C. will remove the trouble. The metal is highly plastic between 300°–400° C.

Finally, a few additional economic uses of calcium may be mentioned. It can be employed as a dehydrator of oils and alcohols, etc., as a means of fixing atomic nitrogen, as a source of pure calcium carbide by direct reaction with pure carbon, and as a stiffening filling for hollow-metal structural members.

Plastic Deformation of Metals

By Richard Saxton

In the cold reduction of metals the results of research on the relation of speed in reduction to quality of production indicates that greater reduction per draft produces higher tensile strength for an equal amount of aggregate reduction.

EFFICIENT plastic deformation of metals by cold working demands a knowledge of crystal flow, and the reaction of a particular metal to cold pressure; a wide variation existing in the response of many metals to the stress imposed by this method of fabrication. Uniform flow under the pressure induced is essential, otherwise cross-sectional stress is set up, leading to a brittle condition of the metal, which is detrimental to further processing.

In cold reduction work by applied pressure, as distinct from deformation in the form of pressings, the reduction schedule is drafted in accordance with the response of a particular metal to the pressure applied. Percentage of reduction per draft is based on the reaction of the metal and aggregate drafts possible before the requisite annealing treatment, if further reduction is essential to finish. Heavy reduction per draft is more the exception than the rule, as this leads to rapid hardening and to less reduction in the aggregate. Lighter stress imposed during the unit draftings renders possible a greater depth of reduction before elastic limit point is reached.

In plastic deformation of steels in wire or bar form, certain fundamental changes take place which are independent of the reduction performed. In the first instance original equiaxed structure is transmuted to a tough, fibrous material. Concurrently, metal is work-hardened and responds less plastically to further work. The original grain formation is also fragmented, and crystallites formed tend to orientate in one specific direction. Cold work effect causes the crystalline structure to assume a practically constant orientation, crystals forming a rhombic dodecohedral plane, perpendicular to the length of metal, and a cube face on surface. Much support is given to the theory that reorientation is the main cause of hardening during cold work.

During the travel of the die-reducing zone, a change in metal structure takes place, due first to slip on the crystal cleavage planes, then by slip between crystals, known as internal and external respectively. Research on this particular reaction reveals that when initial yield has taken place, and the metal has hardened against further yield, there is a proportional decrease in the internal slip, and an increase in external slip.

Dealing with steels as one of the chief metals subjected to cold work in various forms, the higher the carbon and manganese content, the greater the resistance to deformation, both in single and aggregate draftings. Mildness, or extreme ductility under pressure, however, is not a ruling factor, as many soft quality steels, in particular, charcoal iron, are too mild to withstand the tensile stress essential to reduce by an equal percentage per draft, as is possible with steels of higher carbon composition. This is due to the lesser resistance of the mild qualities to tensile stress imposed; if the drafts are too heavy, they have the effect of stretching the reduced section to a much smaller diameter than is furnished by the reducing die.

The effect of stretching, in conjunction with the previous reduction, sets up differential internal stress, which can only be eliminated by prolonged annealing, known in the trade as "soaking," or, with heavy bar material, by machining surface. When employing this method there is always the risk of distortion, due to released stresses adjusting themselves to a state of equilibrium on removal of outer skin.

Plastic deformation of austenitic stainless steels now presents few difficulties, but in the early days of production, when cold work on this particular quality was still in an experimental stage, many problems had to be solved before the technique of economic production became known. One of the most successful methods, ensuring high aggregate reduction percentage, and even flow under pressure, is the lead-coating process. This, as the name denotes, is the application of a lead coat to the steel surface. In addition to facilitating reduction, the coat acts as a lubricant during passage of reducing zone, and is easily removed on completion of the operation.

Application of a coating of softer metal to facilitate reduction is no new discovery, as this method has been employed for a long period in the reduction of many brittle non-ferrous metals, which otherwise would not reduce without the application of heat. Platinum may be cited as one metal resistant to cold deformation in any form in its normal metallic state, which, when coated with silver, is deformed as easily as many metals known for their plastic qualities.

As is well known in the metal trades, the majority of metals necessitate a cleaning or descaling operation before plastic deformation can be performed. This is usually accomplished by immersion in a suitable acid-and-water mixture, hydrochloric or sulphuric acids being the chief agents employed for this purpose. Efficient removal of scale and foreign matter is essential, otherwise there is a possibility of scale left adhering being forced into the surface of the metal during its passage through the die. Where this occurs, a galvanic couple is set up, leading to electrolytic action and resultant corrosion. The scale, being cathodic to the metal surface, is unaffected, but corrosion is set up in the base metal, and continues to spread until the scale is dislodged. A somewhat similar action is set up by pitting or surface defects.

In general, minute traces of oxide or scale, not eliminated by the acid processing, are retained in the pits. During fabrication surface metal flows over these pits or indents, forming slivers or "chibs," resulting in corrosion at these points, and forming, as the slivers corrode through and fall away, a rough, pitted surface.

Research work in relation to speed and to quality of production reveals that greater reduction per draft produces higher tensile strength for an equal amount of aggregate reduction. High speed has no consistent effect on torsional properties, and the best combination of strength, ductility, toughness, and fatigue resistance result from suitable drafting, consistent with the resistance of a particular metal to deformation.

Electro-Deposited Rust Proofing Coatings

Electroplated finishes maintain a position of great importance for rust-resistance in light engineering assemblies, because of their hardness, exceptional wearing qualities, and general durability. Some of the deposits applied for this purpose are briefly reviewed, the choice of which to apply, however, depends upon the service conditions of the part to be protected.

ECONOMY in the use of materials and the avoidance of waste are factors of great importance in normal times, under present conditions, however, they have greater significance. It is sometimes overlooked that the waste resulting from the corrosion of metals is probably the most outstanding cause of waste of these materials, and all methods applied that give protection to metal structures preserve them for a longer useful life and contribute to economy in the use of metals.

Much work has been conducted on the properties of thin films which form spontaneously on metals under certain conditions and protect them from further attack; but here attention is directed to thin electrodeposited coatings which are still of paramount importance for rust-resistance in light engineering assemblies, because of their hardness, exceptional wearing qualities, and general durability. Under present conditions service is of primary importance, and a correct choice must be made to ensure a high standard of protection. The selection depends largely on the conditions of service and the finish given demands a high degree of skill, technical knowledge and experience.

There are two fundamentally different types of electrodeposits on metals. According to Silman,* the first type, of which nickel-plating on steel is an example, consists of a metal that is more "noble" than the metal it is required to protect, i.e., electro-negative or cathodic to the latter. Such coatings fail as a result of their inherent porosity, or when they are mechanically damaged. Once having been initiated at a point, corrosion will proceed more rapidly due to electro-chemical effects. The coating may be lifted locally by the corrosion products formed, so that failure of the plate occurs progressively.

The second type of coatings comprise those metals which are electro-positive to the basis metal, such as cadmium-plate on steel. In these cases the coating, under corrosive conditions, is attached preferentially, so that the basis metal is protected to a degree. Such protection is considered to be active at a little distance, so that corrosion of the underlying metal will not take place even when the coating is porous. The effect of this electro-chemical protection is most marked under conditions of total immersion. It is less evident, however, in normal atmospheric corrosion conditions.

In both types of plating, it is generally recognised that, under commercial plating conditions, by far the most important factor in the production of satisfactory electrodeposits is an adequate thickness of coating; for this reason it is important to specify the thickness. The type of solution to be used is also of great importance, and this, too, should be specified.

Cleaning and Pickling

The entire success of the plating operation depends to a considerable extent on the pre-treatment of the ferrous material. Scale is generally removed by pickling in a bath of about 10% sulphuric or hydrochloric acid. The latter bath is used cold, but the sulphuric-acid bath is more effective at a temperature of about 140° F.; the temperature being gradually raised as the iron content of the bath increases. The strength of the pickling solution should be controlled and maintained by acid additions, but when the iron content reaches about 6% further additions of acid are uneconomical, and the bath should be renewed.

* *Journ. Electrodepos. Tech. Soc.*, 1942, **18**, 21–33.

Alkaline Electro-Cleaning

Oil used in the cold-rolling operations of steel sheet and strip cause films on the surface which give considerable difficulty to the plater. Probably the most effective means of removing these films is by electro-alkaline cleaning. These cleaners consist of combinations of alkalies such as sodium carbonate, or hydroxide, and sodium metasilicate, to which tri-sodium phosphate and soap are sometimes added to facilitate rinsability and emulsification. The work is best made with the cathode, using a high-current density (about 100 amp./sq. ft.), so that strong gassing occurs and the film is broken up and dispersed. Alkaline cleaners are most effective when used hot, and temperatures of not less than 180° F. are generally employed.

Deposited Metals

Nickel-plating is one of the most useful protective coatings on steel, though under war conditions the application of nickel for this purpose is being increasingly restricted. It is especially good when used as an undercoat for chromium, in which case the chromium, as ordinarily applied, has a thickness of only 0·00001 to 0·00003 in. The purpose of the chromium is to prevent tarnishing of the nickel; it adds little to the protective value of the nickel-plate on which the rust-preventive value is almost entirely dependent.

Copper is often applied as an undercoat to nickel, but, generally, thin copper deposits (of the order of 0·0003 in. or less) adds little to the protective value of the nickel top-coat, when the copper is built up to a greater thickness, however, it can replace part of the nickel. Such composite deposits often approach the value of a pure nickel coating, but present no service advantages over the latter. On the other hand, there is some evidence that the protective value of a triple coating, viz., nickel-copper-nickel is greater than that of a nickel coating alone of equal thickness.

Electro-tinned steel is increasing in importance since the advent of the sodium-stannate bath, which enables tin coatings to be obtained of a uniformity and thickness hitherto not producible with the old chloride bath. For outdoor service a minimum tin thickness of 0·0010 in. is recommended, while for mild service conditions 0·0005 in. may be adequate.

Electro-tinning has established itself as a second coating for parts fabricated from hot-tinned sheet; severely drawn or formed components made from hot-tinned sheet are liable to have damaged coatings. In such cases an electro-tin deposit is applied to a thickness of about 0·0003 in. directly on the hot tin.

Electro-tinning is especially useful as a rust-protection on steel where soldering operations have to be carried out on the component, and soldering can be effected on tinned surfaces more easily than on the majority of other coatings.

Zinc and cadmium deposits are perhaps the best known and most universally applicable protective coatings for the ferrous metals. They have several limitations, however, among which is the fact that zinc coatings, when exposed to moist atmospheres, have the tendency to produce a white corrosion product which is liable to exfoliate. Cadmium is less liable to this, so that on moving parts cadmium is superior to zinc, as it is not so likely to lead to interruption of the functioning of the mechanism to which it is applied under conditions of high relative humidity. Apart from this

characteristic zinc coatings are not inferior to cadmium from the rust-protection point of view, and in industrial as opposed to marine atmospheres there is evidence that they are even better.

A process recently developed in the United States, known as Corronising, consists essentially in plating the steel, after alkali cleaning and anodic etching, with an extremely thin nickel deposit followed by a rather greater thickness of zinc and heat-treating the composite coat. Rimbach[1] claims that excellent corrosion resistance is obtained by this method. The heat-treatment is variable, depending on the gauge of the metal coat, the conditions it has to withstand, etc., but a typical treatment would consist in heating the coating for six hours at, say, 350° C. The nickel and zinc coating thicknesses are of the order of 0·000024 in. to 0·000125 in. in the case of the former, and 0·000075 in. to 0·000250 in. in the latter.

Lead-plating has not in the past received any great attention as a rust-proofing coating on steel because of the ready availability of zinc and cadmium, which are generally considered to be superior in this respect. The war has however, led to an increased interest in the metal as a protective for steel in general use, as opposed to its special application where resistance to acids (especially sulphuric acid) is concerned. There are two baths commercially used for lead-plating, viz., the fluosilicate and the fluoborate baths, but the former cannot be used on steel without a copper undercoat, and it is chiefly used in electro-refining processes. The fluoborate bath can be regarded as the only present-day bath of interest to the electro-plater. For the deposition of good lead deposits, up to 0·05 in. in thickness, and for barrel-plating, Gray and Blum[2] recommend the following solution :—

Basic lead carbonate	360 grm./litre.
50% hydrochloric acid	480 "
Boric acid	212 "
Glue	0·2 "
Temperature	77°—105° F.
Current density	10—30 amps./sq. ft.

Anode corrosion is practically 100%, but pure lead anode should be used to prevent sludge formation and " treeing " of the deposit.

Lead deposits of 0·0005 in. to 0·0010 in. in thickness give reasonable protection against corrosion if of low porosity, but owing to the soft nature of the metal breaks in the coating occur relatively easily if it is subjected to abrasion ; lead is cathodic to iron, so that rusting is prone to occur at such breaks more readily than with zinc deposits under conditions of high humidity. With heavier deposits than indicated in the above range, the mechanical properties of the coatings are improved and better durability is obtained.

Special Die Steel for Hot-Pressing Copper-Zinc Alloys

STEELS for dies must be uniformly strong, heat-resistant and heat-conducting, and they must retain their hardness at working temperatures of 450°—650° C. and possess good ductility. The composition of the steels, and the preparation of the dies, must be carried out with great care, states Asemann* in considering die steels for copper-zinc alloy hot pressings.

To the older Cr–Ni steels have been added new Cr–Mo, Cr–Ni–Mo, Cr–W–Mo, and Cr–W–Mo–Co alloys. Some of these are : (1) C 0·38, Si 0·29, Mn 0·50, Cr 1·75, V 0·10, and Mo 0·25%, and C 0·42, Si 0·35, Mn 0·80, Cr 1·95, V 0·20, and Mo 0·70%. These alloys are soft annealed at 680°—700°, hardened in oil at 880°—900°, and tempered at 570°—610°. Their strength is 135—50 kg. per sq. mm. They are suitable for normally used dies. The dies made from these steels are resistant to wear, heat and abrasion. (2) C 0·52 (0·58), Si 0·20 (0·35), Mn 0·50 (0·70), Cr 0·80 (1·00), Ni 1·50 (1·80) and Mo 0·60 (0·80)%. They are soft annealed at 690°—710°, hardened in compressed air blast at 870°—800°, tempered at 530°—600° ; strength is 135—50 kg. per sq. mm. Dies of these heat-resistant alloys are suitable for deep and complicated shapes. (3) C 0·38 (0·42), Si 0·50 (0·75), Mn 0·30 (0·40), Cr 4·30 (4·50), Ni 1·50 (1·70), V 0·15 (0·15) and Mo 0·40 (0·60)%. They are soft annealed 720°—40°, hardened in oil at 930°—50°, tempered at 520°—60° ; strength is 140—50 kg. per sq. mm. These alloys have greater heat conductivity and heat resistance and are less susceptible to volumetric change than the No. 2 alloys. (4) C 0·36 (0·42), Si 0·20 (0·35), Mn 0·30 (0·45), Cr 1·30 (1·60), W 5·20 (5·60), and Mo 0·50 (0·80)%. They are soft annealed at 760°—80°, hardened in oil at 1,050°—70°, tempered 610°—30° ; strength is 135—50 kg. per sq. mm. Dies from these alloys are used for heavy duty ; they are particularly suitable for pressing Cu alloys. They have good ductility and outstandingly resistant to thermal wear. (5) C 0·30 (0·35), Si 0·20 (0·30), Mn 0·30 (0·40), Cr 2·20 (2·60), W 4·00 (4·50), V 0·25 (0·35), and Mo 1·60 (2·00)%. They are soft annealed at 760°—80°, hardened in oil at 1,070°—1,100°, tempered at 620°—40° ; strength is 135—45 kg. per sq. mm. They are used for heaviest duty ; they have highest heat-resistance and sufficiently high ductility. (6) C 0·32 (0·36), Si 0·20 (0·30), Mn 0·30 (0·40), Cr 2·20 (2·70), W 5·00 (5·50), V 0·40 (0·50), Mo 0·80 (1·20), and Co 1·00 (1·30)%. They are soft annealed at 760°—80°, hardened in oil at 1,080°—1,100°, tempered at 620°—70° ; strength is 135—45 kg. per sq. mm. These alloys are used for small dies for pressing pointed edge, and web shapes. Several other alloys are described, and it is emphasised that the alloying and casting of these metals must be as carefully controlled as their subsequent handling.

1 *Metal Finishing*, 1941, July, 360.
2 Electrochem. Soc. Preprint, 80-39, 1941, Oct.
* I. H. Asemann, *Metallwirtschaft*, 1941, **20**, 82-8, 451-8.

North-East Coast Institution of Engineers and Shipbuilders

THE Student Section of the above Institution has arranged a meeting for April 28, at which a paper will be given by S. Mathwin Davis, M.Eng., on " The Place of the Universities and Kindred Institutions in the Professional Training of Naval Architects and Marine Engineers."

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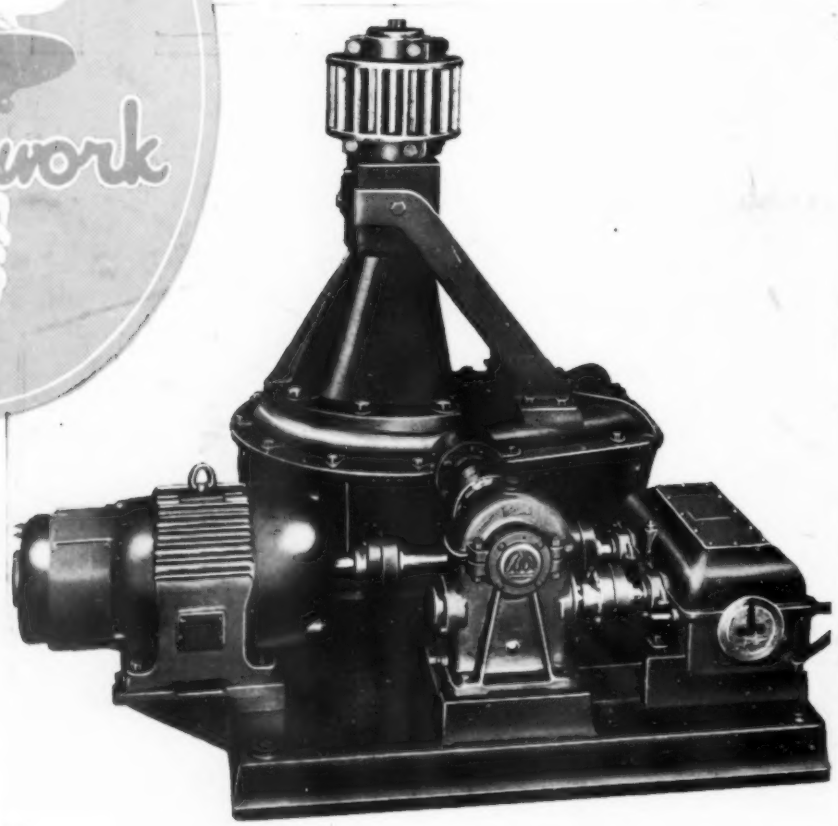

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